FACTORS PREDICTIVE OF ROOF INSTABILITY IN ADDITION TO THE EXISTING CMRR CRITERIA AT TWO CASE STUDY COAL MINES

by

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ABSTRACT

Roof falls remain one of the greatest hazards facing underground coal miners (Barczak et al., 2000; Razani et al., 2013; Oraee et al., 2016). In 2017 there were 91 lost-time injuries from roof falls (in US underground coal mines). A further 48 roof falls were reported in US underground coal mines with no lost days (MSHA, 2018). These numbers have certainly decreased over the last century (MSHA, 2018), but the goal of zero injuries still remains. Assessing the likelihood of roof falls is therefore highly important and will have a direct effect on the prevention of accidents caused by them.

One method developed to help assess roof instability in underground coal mines is the Coal Mine Roof Rating (CMRR). The CMRR is a field-based empirical method which is straightforward to use and gives a quantitative interpretation of coal mine roof geology. The CMRR classification system was developed by Molinda and Mark (1994) to quantify the geological description of mine roof into a single value which could be used in engineering design. It provides an excellent starting point, but it does not necessarily include all the factors that may influence roof stability, nor is it widely used in the Western US. This thesis research uses two underground longwall coal mines located in the Western US (Mine A and Mine B) as case studies to investigate which parameters are indicative of roof falls at these mines. It also evaluates whether the CMRR is applicable to them, and if not, why this might be.

A data set was collected at 30 sites in each mine. This data set included the CMRR, a record of the roof stability and a series of non-CMRR parameters thought to also be potentially indicative of roof stability but which are not included in the CMRR.
These data were then statistically analyzed for correlation between CMRR and roof stability. The correlation between roof stability and the non-CMRR parameters collected was also evaluated. To further evaluate how influential each parameter already included in the CMRR is at each mine, each constituent of the CMRR was removed in turn and a modified CMRR was calculated. This modified CMRR was then evaluated for correlation with roof stability. At Mine A, the correlation between the CMRR and roof stability was found to be statistically significant (significance threshold $\alpha = 0.05$), with a $p$ value of $0.0073$. Logistic regression analyses showed the CMRR to be reasonably predictive of roof stability at Mine A. Faulting, along with depth of cover and slope angle of surface topography were found to be the most significant non-CMRR parameters to correlate with roof stability at Mine A. At Mine B, the correlation between the CMRR and roof stability was not found to be statistically significant ($p$ value $= 0.95$) against a significance threshold of $\alpha = 0.05$. The logistic regression analyses also showed the CMRR to have little predictive capability on roof stability at Mine B. At Mine B, location at an intersection and depth of cover were found to be significantly correlated with roof stability.

The CMRR is therefore moderately effective at Mine A but not effective at Mine B. This is thought to be due, at least in part, to the unusual topography above Mine B, with differential erosion resulting in a landscape of flat plateaus and sharp river valleys. It is suggested that these sudden changes in slope and topography lead to in-situ stress rotation and the development of shear stresses near the excavation at Mine B. This, combined with a lack of major discontinuities such as slickensides, which are central to the CMRR system, likely explains why the CMRR is much less effective at predicting
roof stability at Mine B compared to Mine A. Mine A was also found to more closely match the geological conditions of the mines in the CMRR reference database than Mine B. The majority of the coal seams sampled in the CMRR reference database are located in the Appalachian or Illinois basins. The Appalachian Basin is a foreland basin with a complex geological structure and a high incidence of faulting. This is similar to the regional geology at Mine A. The Illinois basin as a whole more closely matches the geological setting at Mine B; both are located in broad, gentle structural depressions. However, the Illinois Basin coal seam most frequently sampled in the CMRR reference database is one with notable faulting and a roof geology which is complex and laterally inconsistent. This is the opposite of the geological conditions in the roof at Mine B, which are laterally uniform and continuous. It is likely that the CMRR is not applicable at Mine B because the geological conditions there are not captured in the CMRR reference database.
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1.1 Background and Problem Statement

Roof fall is considered to be one of the most significant hazards in underground coal mines (Mark et al., 2004; Palei and Das, 2009; Oraee et al., 2016). Hundreds of roof falls still occur every year (MSHA, 2017) and many of these still cause injury and lost time as well as damaging equipment, disrupting ventilation, and blocking emergency escape routes. Even though a single roof fall event might cause fewer injuries or fatalities than a gas explosion for example, the total damage and overall number of injuries caused by roof falls are greater than any other type of mine accident (Deng et al., 2014; Harris et al., 2014; Wang et al., 2018).

One method developed to help predict roof instability in underground coal mines is the Coal Mine Roof Rating (CMRR) (Molinda and Mark, 1994). There are several other approaches for evaluating underground roof stability, some of which will be discussed in Chapter 2.12. The CMRR is a field-based empirical method that is straightforward to use and gives a quantitative interpretation of coal mine roof geology. This method is simple and has the potential for being widely applicable and useful. However, it does not appear to be widely used to predict coal mine roof instability, particularly in the Western US.

Two Western longwall coal mines (Mine A and Mine B) were visited to develop a hypothesis and research plan evaluating the potential use of CMRR in roof stability prediction in Western coal mines. The initial observations were as follows:
Mine A

- Significant lateral geological variation in the roof strata was observed.
- Moisture sensitive mudstone was present at various locations throughout Mine A.
- Sandstone channels and slickensides were observed. The mine geologists at Mine A have mapped the sandstone as channels with a lateral extent spanning the entire mine.
- It has been informally observed by the mine engineers that where the sandstone channel is present in the immediate roof, the roof is less stable.

Mine B

- Mine B was observed to have laterally continuous roof geology and stratigraphy.
- There were still areas with stable roof and areas of unstable roof (i.e. the roof stability was not uniform even through the roof geology was).
- The engineers used the depth of cover, the slope of surface topography and the presence of the coal seam merge to assess roof instability and inform their support plan.

Neither mine used the CMRR to assess roof instability. The mine engineers at both mines knew about the CMRR but seemed to find other methods to be more useful for their mine. They suggested that the CMRR criteria did not apply or were not going to give them enough value to be worth their time and effort in calculating it. One of the engineers at Mine A also thought there would likely be several (approximately 5 or
more) different CMRR values across the mine; although this would potentially allow a map of CMRR ranges to be made and roof support decisions made accordingly, the extra effort required to perform such an investigation was not considered commensurate with the expected benefit in terms of roof stability assessment.

These observations led to the development of the following problem statement and hypothesis:

Problem Statement: What geological and geotechnical factors in addition to the existing CMRR criteria could also be predictive of roof instability?

Hypothesis: The parameters currently considered by the CMRR do not fully explain roof instability. Where the input parameters currently considered by the CMRR remain relatively constant i.e. the immediate roof geology does not change laterally, there are other factors that are correlated with roof instability in underground coal mines and which could potentially be included in the CMRR.

1.2 Goals and Specific Objectives

The overall goal of this research is to investigate if there are other parameters indicative of roof fall potential at the two case study mines that are not included in the CMRR, but that potentially could be. Specific objectives of this thesis research are:

1) Evaluate which non-CMRR parameters are correlated with roof instability at Mines A and B

2) Evaluate whether the CMRR is or is not applicable at these mines and if not, why this might be.
3) Provide a starting point for future work incorporating more mines with a view to improving the existing CMRR system.

1.3 Overall Approach

CMRR measurements were made in the field following the methods of Molinda and Mark (1994) along with detailed notes on the roof stability and an assessment of potential factors influencing roof stability. Depth of cover and slope grade data were obtained from mine scale maps. Faulting and large-scale features such as sandstone channels were recorded through observation and information from core logs, if available. Once these data were collected, the information was evaluated to see whether there were locations with the same CMRR values but different classifications of roof stability. This was evaluated using the Analysis of Variance test (ANOVA) and logistic regression. Correlation between each variable outlined above, such as depth of cover or the presence of faulting, and the CMRR or roof stability was then evaluated. Throughout the analyses, different approaches to roof stability categorization were considered. Initially, a binary classification of stable or unstable was used, but because in reality an infinite number of degrees of relative stability could theoretically exist, using a finer scale was deemed more appropriate. Based on the level of detail in the data recorded, a scale of 1 – 4 was used, with 1 being the least stable and 4 the most stable. The analyses were run using both the binary and scale classifications of roof stability.

If the current CMRR successfully predicts roof stability at a given mine, e.g. the stable roof has high CMRR values and the unstable roof has low values with no overlap, then no additional variables need to be added or analyzed. However, it is expected that
there will be a spread of CMRR values for each roof stability category. Such a
distribution would then imply that the parameters currently included in the CMRR do not
fully explain roof instability, leading onto further investigation of what the additional
parameters might be. To further analyze how effective the CMRR is at the two case
study mines, the CMRR values at each site were recalculated with all but one of the
component parameters, e.g. without the moisture sensitivity parameter or without the
cohesion-roughness parameter. These adjusted CMRR values were then tested for
correlation with roof stability.

1.4 Thesis Organization

This thesis has been organized into seven chapters as described below:
Chapter 1: Introduction and Problem Statement. This chapter provides an outline of the
problem and the approach as well as the goals and specific objectives of this research.
Chapter 2: Literature Review. This chapter discusses the development of the CMRR
and its use both within the United States and internationally. A discussion of parameters
also thought to be indicative of roof stability but not included in the CMRR is also
provided, along with an outline of other methods used to assess roof stability in
underground coal mines.
Chapter 3: Case Studies. This chapter provides a summary of the mine geology, with a
focus on ground control, at the two case study mines used in this research.
Chapter 4: Methods. This chapter provides a detailed description of the methodology
used for the research. The methods of fieldwork and data collection are described, as
are the methods used to analyze the data collected.
Chapter 5: Results. This chapter presents the results obtained from carrying out the analyses described in Chapter 4.

Chapter 6: Discussion. This chapter evaluates the results in the context of the research hypothesis. A discussion of assumptions, uncertainties and limitations is also provided.

Chapter 7: Summary, Conclusions and Recommendations. This chapter presents the research conclusions, a research summary and recommendations for additional research.
2.1 Introduction

This chapter has been subdivided into different sections based on their relevance to the thesis topic. These sections include: a history of the CMRR and how it is used in practice, the advantages and disadvantages of using the CMRR, and examples of its use both within the United States and internationally. Additionally, a section summarizes parameters also thought to be indicative of roof stability but which are not included in the, along with an outline of other methods used to assess roof stability in underground coal mines.

2.2 Development of the CMRR

The engineering properties of rock can be highly variable from mine to mine. While geologic reports contain much descriptive information, from an engineering design standpoint, quantitative data are required. More specifically it is the strength of the overall rockmass rather than that of a small specimen which is required. Rockmasses are unique in that their strength can seldom be determined accurately (Mark and Barczak, 2000; Whittles et al., 2007). Directly measuring the mechanical properties of a rockmass in-situ is generally not possible, due to the cost and difficulty of access to the different strata in the immediate roof (Whittles et al., 2007). Rock strength has therefore been, and still is, traditionally estimated from laboratory tests. The uniaxial compressive strength test is commonly used but the triaxial compressive strength test
more accurately simulates the 3D stress a rock will encounter underground. Shear tests of discontinuity planes, in particular bedding, can also be useful to evaluate the likelihood of slip between roof layers, however they are not commonly performed in the US (Mark and Barczak, 2000). Rockmass classification systems such as RQD, RMR and Barton’s Q system were developed to help quantify the strength of a rockmass (Deere, 1964; Barton et al., 1974; Bieniawski, 1989). They have been widely used in industry as they are straightforward and practically useful: they provide a simple method to characterize rock strength and bridge the gap between geologic descriptions and engineering data. They also make it possible to compare ground control between sites, even where the geology is notably different.

These classification systems were designed empirically and based on a large number of case histories. However, when detailed design criteria for a specific engineering application are needed, these systems can be too general to be useful (Whittle et al, 2007). They are not especially effective in coal mines as they are primarily focused on jointing, whereas bedding is generally the most influential discontinuity affecting coal mine roofs (Molinda and Mark, 1994). Systems such as RMR also rate rock units individually, but coal mine roofs are generally comprised of several layers of varying character (Mark and Molinda, 2005; Calleja, 2006). There is also variation between the stability requirements of mines and other underground excavations such as tunnels, not least due to the difference in expected life span for the respective developments.

The Coal Mine Roof Rating (CMRR) is an empirical method developed in the 1990s by Molinda and Mark (1994) to bridge the gap between geologic variation in
underground coal mines and engineering design. It generates quantitative data on the in-situ rock and is based on the idea that the structural competence of mine roof rock is governed primarily by the discontinuities in the rockmass. It was designed to be straightforward to carry out and easily repeatable as well as to provide meaningful predictions of roof stability. The CMRR is based on a large quantity of real-world case histories, analyzed statistically to identify the most important factors in assessing roof stability (Molinda et al., 2000). It was designed specifically for coal mine roofs and to fill the gap where more traditional rockmass classification systems were less applicable. The RMR system is therefore not considered an alternative to the CMRR necessarily, but the CMRR was designed to be equivalent, such that the unsupported span-stand up time relationship should be almost the same in both systems (Mark, 1999).

The first step taken in developing the CMRR was to identify which rockmass attributes were significant in quantifying roof rock competence. Their relative importance was also evaluated. This was done by drawing on coal mine ground control experience from several decades previously (Mark and Molinda, 2005). This information was broadly split into two categories; information on specific geological features and studies done generalizing results for specific mines, regions and countries. A variety of geological factors such as bedding, moisture sensitivity and slickensides were considered to be influential for ground control in underground mines by various different authors (Newman and Bieniawski, 1986; Buddery and Oldroyd, 1992; Molinda and Mark, 1994).

Bedding was the factor most widely agreed on as causing ground control issues in underground coal mines. In particular, weak laminations and thin interbeds of
sandstone and shale were considered to be the most problematic – common to both are closely spaced weak bedding planes. Slickensides and minor discontinuities were also considered important so Molinda and Mark (1994) decided to include a ‘multiple discontinuity adjustment’ in the CMRR. The next question in the development of the CMRR was how far up into the roof should the rating extend and how to combine the properties of the layers into one single output. Mark and Molinda (2005) considered that bolt length essentially determined the thickness of the mine ‘roof’ and that the influence of the layers above the bolts could be treated as negligible. Experience also showed that the best scenario is when roof bolts anchor in a strong layer. Moisture sensitivity was also widely agreed to cause problems in coal mine ground control (Van Eeckhout, 1976) and the presence of water was proposed to be included as an adjustment factor (Molinda and Mark, 1994).

Large scale (approximately 10s of meters scale or greater) features such as faults and sandstone channels were not included in the CMRR, although it is suggested that separate CMRR values can be developed for ‘normal conditions’ and others for areas where large scale features are present (Mark and Molinda, 2005). Such large-scale features are considered to be anomalous conditions and to need specialized support (Molinda et al., 2000). Mark and Barczak (2000) agree, saying these features should be treated separately as they can cause significant disruptions in relatively small areas.

The CMRR was initially developed to be calculated from field observations at overcasts and previous roof fall sites as well as surface highwalls (Molinda and Mark, 1994). A few years later a method for assessing the CMRR from drill core was also
developed (Mark and Molinda, 1996). In the 2000s a computer program with an AutoCAD interface was specifically designed to calculate the CMRR. This program is available to be downloaded for free from the NIOSH website. Calleja (2006) also developed a rapid rating system to calculate CMRR values for large quantities of drilling data, essentially calculating CMRR for multiple drillholes at the same time.

2.3 CMRR Input Parameters

The current CMRR rating system includes the following parameters:

UCS: Laboratory testing is considered to be the standard way to determine the Uniaxial Compressive Strength (UCS), however, the Point Load Test (PLT) is considered to be an acceptable alternative. If core is unavailable, and/or the measurement is being taken underground, the UCS may also be estimated by an indentation test as proposed by Williamson (1984). This is where the rock is struck with the round end of a ball-peen hammer and the impact reaction compared to the CMRR reference chart.

Discontinuity Intensity: This refers to both the spacing between discontinuities and their persistence. These are both measured directly using a rating scale and observations – the output of this is a value which is added to the overall rating. If core is available, the RQD and/or fracture spacing may be calculated and used in a series of equations to calculate the discontinuity rating value. This input parameter was designed such that widely spaced persistent joints would give a notably different rating than narrowly spaced less persistent joints. This is important because these two styles of jointing will behave differently in practice.
Shear Strength of Discontinuities: Based on the understanding that the greatest loading in a coal mine roof is lateral (due to horizontal stress), this rating considers the cohesion and roughness of discontinuities. Cohesion is estimated using a mason chisel and hammer to split samples of rock. The number of chisel blows required to split a sample of rock along bedding planes may be correlated with the cohesion. Surface roughness is estimated visually using a comparison system based on that introduced by Barton et al. (1974). This roughness scale is intended to be applied on scales from hand sample size to several feet across a fall exposure. If core is available, then the diametral PLT may be used to estimate shear strength based on a conversion table (Mark and Molinda, 1996).

Moisture Sensitivity: This is included in the CMRR as a deduction of points. It is usually calculated by means of an immersion test, but if this is not possible, it is sometimes estimated visually underground. Given the potential lag time for humid mine air to affect rock strength, the CMRR is sometimes reported with and without the deduction for moisture sensitivity. This parameter includes the effects of water, albeit indirectly.

The CMRR unit rating is then calculated using the following equation:

\[ \text{UCS rating} + \text{Discontinuity Shear Strength Rating} + \text{Discontinuity Intensity Rating} + \text{Moisture Sensitivity Deduction} = \text{CMRR Unit Rating} \]

This calculation is done for each unit in the bolted interval or immediate roof. Next, the thickness-weighted average from each unit in the roof within the bolted
interval is calculated. Several adjustment factors are applied to this weighted value. These are described below.

Strong Bed on Bolted Interval: The stability of mine roof tends to be calculated by the strongest bed in the bolted interval (Mark and Molinda, 2005). The prominence of this influence depends on how much stronger the bed is compared to the other units. It also must be > 0.3 m thick to be considered to provide extra support and the bolts must be anchored in at least 0.3 m of the bed. The adjustment factor itself is calculated by an equation derived using multiple regression (Mark and Molinda, 2005).

Number of Units: Mine roofs containing multiple units have been observed to be less stable than those composed of a single homogeneous rock type (Mark et al., 2004). This adjustment factor takes the form of a points deduction which increases as the number of weak contacts increases.

Groundwater: The RMR rating scale is used for this adjustment, with a maximum deduction of 10 points for flowing water present. If water is present at the CMRR measurement site, it is accounted for in the final value. If there is no water present, this adjustment factor is not included.

Surcharge: This refers to the strength of rocks overlying the bolted interval. They are only considered when they are significantly weaker than the rocks within the bolted interval. This is again accounted for by a points deduction in the CMRR rating.
2.4 CMRR Output

The CMRR outputs a single number from 1-100. The lower the value, the less stable the roof. In practice, most CMRR values are above 25; lower than this and the roof is likely to collapse immediately upon mining (Molinda et al., 2001), although this is certainly not always the case.

Molinda and Mark (1994) suggest the following categories of roof competency based on CMRR values:

CMRR < 45 = Weak Roof

CMRR 45-65 = Moderate Roof

CMRR > 65 = Strong Roof

At the time of the original research (Molinda and Mark, 1994), it was found that 76% of the data from Southern Appalachia collected in the CMRR development process fell into the weak or moderate category.

2.5 Using the CMRR

The input data used to calculate the CMRR may be obtained from underground exposures or from drill core. The rating system was also designed such that a mine engineer could use the system and “make the most of the information available” (Mark, 2018). In many mines, the location in which the engineers are trying to assess stability may not have any previous falls or exposures showing the full profile of the immediate roof. In such cases, inference may be used along with knowledge of the mine geology. The lowermost unit can also be tested in these cases, although its thickness will be an
estimate unless core was taken at the same location (Mark, 2018). Osouli and Shafi (2016) ran into this issue where there was no underground exposure of the rock layers above the 3 ft of immediate roof. They calculated the CMRR for the immediate 3 ft and used borehole information and inference to evaluate a suitable CMRR value for the other 3 ft of the bolted interval.

In both the underground and drill core methods, UCS, discontinuity spacing & persistence, discontinuity cohesion & roughness and moisture sensitivity are measured and input in the CMRR unit rating calculation. The underground data collection sheet for the CMRR is shown in Figure 2.1.

The first step in calculating the CMRR is to divide the mine roof into structural units, which are essentially layers that have similar engineering behavior. The CMRR is then calculated by weighting all of the unit ratings within the bolted interval by their thickness and applying the appropriate adjustment factors.

2.6 CMRR Drill Core Methods.

Osouli and Shafi (2016) found the CMRR values calculated underground and those calculated using drill core showed reasonable agreement, although they only used one mine in the Illinois basin in their study. No other studies comparing the two methods were found.

Where core is available, the roof may be divided into lithological/structural units from the core logs and the UCS obtained from point load or UCS tests of the core. Next, the RQD and fracture spacing are calculated and used to calculate the discontinuity persistence and spacing. Both underground and with core available, the cohesion
Figure 2.1: Underground data collection sheet for the CMRR (Mark and Molinda, 2005).
is estimated using a chisel and compared to a rating scale. Surface roughness is estimated by visual comparison with rating charts in both cases. If core is available and disposable, the moisture sensitivity may be estimated through an immersion test, in the same way as for samples collected underground. The presence of a strong bed on a bolted interval may be estimated from core or logs, and the corresponding adjustment factor applied. Similarly, the number of units should also be estimated from core and logs, if available. The groundwater adjustment should be based on observations underground, but the surcharge effect can be estimated from the core.

2.7 CMRR Application in the US

Since its introduction in the 1990s, the CMRR has been used extensively in the US in the Analysis of Longwall Pillar Stability calculation (ALPS) (Mark and Molinda, 2005), although records of its use in the prediction of roof stability are few. The ALPS output is a safety factor which varies with the strength of the roof rock. The Analysis of Roof Bolt Systems (ARBS) also uses the CMRR as an input. It is also possible that some mines in the US do use CMRR to assess roof stability without publishing any record of doing so. One published example of the CMRR being used in the design process is that by DeMarco (1994), who used it to help design the yielding pillar gate road in an Eastern US longwall mine.

From the literature there is on CMRR, it is apparent that its use in the US has been primarily in the Eastern US, particularly in the Illinois and Appalachian coal basins. If the CMRR has been used in the Western states, there has been no record made in the literature. It is currently not known whether the CMRR is less applicable in the West
or if it has just seen limited use there. This could be due to differing geological conditions or different factors controlling roof competence in the Western US, which are not part of the CMRR.

Osouli and Shafii (2016) used the CMRR to evaluate the roof condition and support design for an underground coal mine in the Illinois Basin. They found the CMRR to be a reasonable method for classifying the roof rockmass in the Illinois basin, but they do note that the CMRR was based on data mainly from Appalachian coal fields which typically have more competent roof compared to elsewhere in the US. Likely for this reason, Osouli and Shafi (2016) admit that the application of the CMRR in some other coal fields has encountered difficulties and its use has been limited elsewhere, although they did not find this to be problematic when using the CMRR on the weak, moisture sensitive shales of the Illinois basin. Calleja (2006) also notes the CMRR should be used with caution in mines which have different rock characteristics from those used in the original CMRR development.

Pappas et al. (2000), examined the roof fall injury rates in longwall mines from 1995-1998 (using MSHA data) throughout the United States and found that the rate in the Western US was lower than the national average. This is illustrated in Figure 2.2.

They also examined the rates of non-injury roof falls between 1995 and 1998 and found them to be lower in the Western US than in Western Kentucky, Illinois and Indiana. However, when they looked at roof and rib fatalities, Western US longwall mines showed a contrastingly high value. This is shown in Figure 2.3. With that in mind, it is worth noting that a period of three years will likely not give a fully representative trend and also that by the 1990’s the number of fatalities per year due to roof fall
was very small (<10) (MSHA, 2018), meaning that a single additional value could skew the data significantly. Pappas et al. (2000) also note that high rib fall rates occur in 80% of Western longwall mines, although they do not define what is meant by high.

It is therefore highly possible that a rib fall caused the Western US value to increase in Figure 2.3, although roof and rib stability are arguably related. The data in Figures 2.2 and 2.3 should therefore be treated with caution.

NIOSH has also been measuring CMRR in its ground control studies for nearly two decades and their database includes several hundred observations. It is unknown where these observations were made and what percentage of them came from Eastern versus Western US states. From these data however, Pappas et al. (2000) have shown
that nearly all of the low CMRR (<35) measurements were made in the Appalachian or Midwest coal basins. This could be an indicator of why the CMRR is less commonly used in the West; there may simply be less need for it.

![Figure 2.3: Roof and Rib fatality rates by location, 1995 to 1998. Source: MSHA Data. (Pappas et al., 2000).](image)

In 1994, during the CMRR development process, the CMRR was calculated for 97 roof exposures at 75 coal mines throughout the United States (Molinda and Mark, 1994). Of these exposures, four were from Colorado, five from Utah and one each from Wyoming and New Mexico respectively, equating to a total of 11 mines or 11% of the total 97 exposures being located in the Western US. Molinda and Mark (1994) do acknowledge this potential bias to some extent by saying the distribution by region of the data...
approximately reflects the “extent of mining activity in that region”. The range of CMRR values observed in the US in this data set of 97 mines collected in the initial CMRR development process (Molinda and Mark, 1994) is shown in Figure 2.4.

From Figure 2.4, Molinda and Mark (1994) found a high percentage of coal mines roofs in the Northern Appalachian region (95%) to be classified as weak or moderate. In contrast, they found most of the CMRR values from Utah to be classified as strong on the graph in Figure 2.4. They note that this supports ‘past observations’ that coal mine roof rocks in the Western US are stronger than those in Eastern US mines, although they acknowledge that these are preliminary observations and that they are awaiting a larger database to confirm this. No such larger database has been published.

Figure 2.4: Range of CMRR values observed in the US (Mark and Molinda, 1994; Mark and Barczak, 2000).
Pappas et al. (2000) also carried out an evaluation of the effects of seasonal patterns including temperature, barometric pressure and humidity on roof fall rates. Due to arid climates with minimal fluctuations in humidity, however, Colorado, New Mexico, Utah and Wyoming were not included in that study. It is therefore implied that the effects of seasonal humidity in these states on roof fall rate is negligible and the probability of roof fall is reasonably constant year-round. This would in turn suggest that moisture sensitivity is less relevant in the prediction of roof fall in these locations. This could be another possible reason why the CMRR is used less in the Western US – the parameters currently included in the CMRR may be less relevant there.

2.8 CMRR Application Outside the US

Interestingly, there are more records of the CMRR being used overseas than in the Western US. In Canada, the CMRR was used in an Underground Coal Mine Safety Research Consortium project aimed at establishing the best practice for conducting geological and geomechanical assessments. The CMRR allowed the Canadian mines to be compared with each other and with international benchmarks (Mark et al., 2002).

In the same paper, Mark et al. (2002) also describe how the CMRR was used in a research project sponsored by the Safety in Mine Research Advisory Committee in South Africa. The project investigated the causes of fatal roof failures in South African coal mines. Data was collected at 182 roof fall sites and it was found that roof falls were more likely where the CMRR was lower, although no other details are given. Butcher et al. (2001) found the CMRR easy to use and suitable for adequately describing the roof conditions at most South African coal mines. They found it took only four hours for a
trained geologist to become familiar with the CMRR and able to start collecting data. Butcher et al (2001) did however suggest some adjustments to the CMRR – notably for joint orientation, blasting and horizontal stress, although they noted that the CMRR results were more reasonable than those from the RMR system which consistently overrated ground conditions. No further evidence of CMRR use in South African coal mines has been found, however.

The CMRR has also been widely used in Australia for mining method selection and mine roof design (Calleja, 2006; Hill, 2007). Australian coal industry research in the 1990s found an average CMRR of 50 at Australian longwall mines and 86% of the data collected fell into the ‘weak’ or ‘moderate’ categories as outlined in Chapter 2.2 (Colwell, 1998). Compared to an average of 75% of US mines falling into these categories (Molinda and Mark, 1994), this might imply lower roof competencies in Australia, although this is difficult to generalize. In a later study, Colwell (1999) used the ALPS system as a starting point for an Australian coal industry research project aiming to develop an Australian chain pillar design methodology. ALPS safety factors and the corresponding input CMRR data were calculated and calibrated to the different geotechnical and mine layouts in Australia. Colwell’s study (1999) found strong relationships between the CMRR, the tailgate SF and the installed level of primary support. The design equations they developed reflected this and the final outcome, Analysis of Longwall Tailgate Serviceability (ALTS), has become widely used in Australia.

Until this year there has been limited documented use of the CMRR in Chinese coal mines. Recently, however, Wang et al. (2018) investigated the application of the
CMRR in two Chinese coal mines. They calculated the CMRR in 11 locations at the two coal mines and found the chance of roof failure to be very low if the CMRR was >50, provided adequate support was installed in the mine. They concluded that the CMRR was useful in the preliminary investigation of stability in the two Panjiang coal field mines that they used as case studies. The roof fall rate and fatality rate from roof fall is much higher in China compared to the US; in 2012, there were 366 roof falls and 459 associated fatalities in China (Wang et al., 2018). The CMRR could therefore prove to be a very useful tool in assessing the likelihood of roof fall in Chinese coal mines provided it is widely applicable and relevant to the conditions encountered there.

There does not seem to be any other published evidence of the CMRR being used in Asia or elsewhere in the world. This is likely because it is a US-developed system and is an input in the ALPS calculation which is commonly required of mines by the Mine Health and Safety Administration (MSHA). Where there is no legal reason to use the CMRR, there is perhaps not any other motivation to use it. This again relates back to the question of its applicability and completeness. Perhaps, if it were more broadly relevant and could provide useful information on roof stability at a wider variety of mines, mines would be more likely to use it for the reason it was developed: to predict roof stability and inform their support plan.

2.9 CMRR Advantages

It has been argued that CMRR is the best available predictor of roof fall rate (Mark and Molinda, 2005): low values (<30) were found to correlate with high roof fall rates and high values (>60) were found to correlate with strong mine roof (Molina et
al., 2000). Further, an advantage of the CMRR is considered to be the inclusion of critical variables, which may be difficult to measure, through the use of rating scales (Molinda et al., 2000).

Geologic features such as slickensides, clay veins and ancient stream channels have been closely linked with many groundfall fatalities (Pappas et al., 2000). Those authors found slickensides to be the primary cause of failure in a series of groundfalls which led to four fatalities and seven injuries; the falls occurred above the permanent support. Failures from geologic discontinuities commonly occur suddenly and with little warning. In this respect, the CMRR focus on discontinuities is an advantage, particularly where the roof stability is governed by geology. Whittles et al. (2007) agree that focusing on discontinuities is beneficial in certain situations. For example, if a failure is governed by a particular joint set with lower shear strength, this is what will determine the rockmass behavior not the characteristics of the rockmass as a whole. In such a case, defining the rockmass itself is not especially useful, since the discontinuities are the primary influence.

The focus on the structural competence of the roof with an emphasis on bedding also means the important predictive factors of skin failure are captured by the existing CMRR. Skin Failure is considered to be the fall of rock blocks less than 2 ft thick from between the bolts or primary supports in the mine roof. The most common causes of skin failure are thought to be the incompetence of the roof strata and the presence of geological discontinuities (Bauer and Dolinar, 2000). Mark et al. (2001) agree stating that particularly in strong, massive roof cracks, joints, crossbeds, and slickensides are the most likely causes of skin failure. Nonetheless, the specific geologic and stress
conditions associated with skin failure are considered to need further investigation (Bauer and Dolinar, 2000). The current CMRR may then require some additional considerations once these conditions are better understood if it is to be used as a tool to predict skin failure as well as larger scale roof instability.

The CMRR has also proved to be a useful input in the ALPS classification system as a method of quantifying rock strength. Mark et al. (2002) showed that when the roof was strong (CMRR > 65), longwall pillars with an ALPS Safety Factor (FS) as low as 0.7 could provide satisfactory tailgate conditions. When the roof was weak however (CMRR < 45), the ALPS SF could need to be as high as 1.3 (Mark et al., 2002). Further, the CMRR can be a useful way to summarize site geology, for example a contour map of CMRR is a more efficient and simple presentation than multiple maps showing different parameters such as UCS, lithology, RQD etc. (Calleja, 2006). The CMRR also allows ground control and rockmass strength to be compared between sites and can be a useful tool to highlight rockmass strength variability.

Roof geology is extremely important in ground control, and the CMRR makes it possible to quantify this at least to some extent and gives a repeatable and meaningful measure of roof quality. However, it is by no means comprehensive, although this is arguably not possible given the vast range of possible geological conditions encountered in underground coal mines.

According to Molinda et al. (2001), there were 790 injuries and 13 fatalities due to roof fall in the US in 1998. From 2006-2010 however, there were only 26 fatalities from roof fall in US coal mines (Harris et al., 2014). Wang et al. (2018) attribute this improvement to the use of the CMRR in roof hazard assessment, pillar design and
modeling inputs. However, while the CMRR surely played a part, it seems likely that other ground control improvements also contributed to this improvement in safety.

2.10 CMRR Disadvantages

The CMRR is designed to focus on the specific features which are common to coal measure rock (Mark and Barczak, 2000). Calleja (2006) agrees that it weighs: “some of the geotechnical factors which may affect the competence of mine roof”. The words ‘some’ and ‘may’ imply that there is still significant uncertainty associated with the system and again that it is not fully comprehensive in assessing roof instability.

Further, Calleja (2006) cautions that the CMRR (like many rockmass classification systems) may be appropriate for its original application – in this case for conditions similar to those of the case studies from which it was developed – but may not work well in cases outside the database. Indeed, the CMRR was designed to apply to all US coal mines (with a potential bias to the Eastern states), but this could mean it is less applicable for use abroad. Wang et al. (2018) agree, stating horizontal stress is not weighted strongly enough in the CMRR to make it widely applicable in China. They attribute this to the CMRR being a US-derived system and in the US, the horizontal stress is generally not much different from the vertical stress (Hill, 2007). The Illinois basin may be a notable exception to this however. Ingram and Molinda (1988) found the maximum horizontal stress in Southern Illinois could be as much as three times the vertical stress. Calleja (2006) also notes that the CMRR gives an indication of rockmass strength rather than rockmass stability, although the two are undoubtedly related.
In studies seeking to use the CMRR to identify locations that require extra ground support, it is important to know the thickness of the weak strata. However, surface borings are rarely at sufficiently close spacing to fully define the thickness of a weak unit. Inference is required, introducing uncertainty. Inconsistencies in human judgement between those recording the CMRR at different locations also introduces error. As a subjective empirical system, there is a high likelihood of variation in results between those making CMRR measurements. Although the CMRR is considered reasonably uncomplicated, Hill (2007) agrees that there is still a high human error factor in its measurements. In particular, he notes that it is common for a reasonably uniform roof material to be logged as a single lithological unit when in reality, the structural competence of the unit can vary significantly. Valuable information can be lost unless the logger is aware of the level of detail required to confirm the homogeneity of the unit.

The CMRR does not account for pre-existing or mining induced stresses, nor does it include mining geometry such as the span or orientation of mine workings. It also does not include the rib or floor features which could potentially continue into the roof. Calleja (2006) therefore concludes that the CMRR alone is insufficient for determining geotechnical design parameters such as roof support, roadway span, pillar design. Indeed, the ALPS database uses CMRR as an input to evaluate longwall pillar stability, in addition to other factors such as horizontal stress. Hill (2007) also does not consider the CMRR to be a “perfect” rockmass classification system: in Australia, the CMRR is by no means used exclusively and is strongest when used in conjunction with other methods. This is often the case for rockmass classification methods: the best outcomes are obtained when results from multiple methods are compared and cross
checked. The CMRR as it is currently, is therefore perhaps best used ‘holistically’ in
technical characterization and mine design (Hill, 2007). Further, the current CMRR
cannot be used to directly design support measures in underground mines as it does
not include the width of roadway, roof density, or the bolt orientation (Wang et al.,
2018). They therefore suggest the CMRR might benefit from some modification or
inclusion of additional parameters. The CMRR also does not directly include the effect
of strength decay over time (Osouli and Shafi, 2016).

It is also worth noting that in a typical underground coal mine, geology can vary
over short distances. Without systematic coring, it is difficult to know whether one
CMRR value accurately describes large sections of the roof. In some ways this
ambiguity makes the system less straightforward or easily comparable between mines.
This could be resolved to some extent if a mine were to be described by a range of
values or one general number with a second for notably weak or strong areas. Molinda
et al. (2000) suggest a mine may be delineated into zones with common variables and
similar geotechnical environments.

2.11 Addition of Parameters to the CMRR

According to Griffith et al. (2014), roof stability in underground mines is controlled
by the quality and thickness of the rock layers, the excavation geometry, the stress state
around the excavation and pre-existing geologic features like faults, joints and sand
channel deposits. Some of these parameters are currently included in the CMRR and
some are not. Although their work was focused on stone mines, Esterhuizen et al.
(2011) agree, noting that observed roof instabilities in underground mines are generally
due to excessive horizontal stress or unfavorable geological structures which lead to block fall or beam failure along bedding planes.

Further parameters which are not currently included in the CMRR but which are also thought to influence roof instability are discussed in this section. Faulting (both large and small scale) is also discussed as it is highly prevalent at Mine A. As Calleja (2006) says, improving the CMRR will benefit the industry if the results become simpler to obtain and if more information such as pre-existing drilling data can be used. Wang et al. (2018) are also in favor of continuing work that will make the CMRR more robust and comprehensive and suggest it be expanded to include case studies from China and other countries.

Taheri et al. (2017) experimented with adding two additional parameters to the existing CMRR – width of gallery and density of overburden rock. They were motivated to modify the CMRR because it currently cannot be used directly to design support measures in underground mines. Using the Eliza Hill project in Australia and the Tabas coal mine in Iran as case studies, they evaluated the correlation between CMRR and Factor of Safety (FS) as well as carrying out a parametric study on various rock properties and gallery geometries. After adding two additional parameters to the CMRR (width of roof span and density of overburden rock), they used the modified system to recommend a suitable bolting system, including length and spacing. This was then verified by numerical modelling. Taheri et al. (2017) found the support requirements predicted from their modified CMRR system to match almost exactly with the numerical analysis results. Aside from that by Taheri et al. (2017), no other studies attempting to adjust or add to the CMRR have been found.
The following parameters are also considered to be potentially indicative of roof stability in underground coal mines. They are discussed in reference to the CMRR as it is currently. This list is not exhaustive; rather, it is intended to give an overview of some parameters which could also correlate with roof instability over and above those already included in the CMRR. Griffith et al. (2014) as well as Molinda and Mark (1994) state that roof instability can be divided into geologic and stress related mechanisms (and also post-mining rock degradation from fluid exposure).

2.11.1 Stress Field

The CMRR does not currently include horizontal or vertical stresses in its calculation explicitly. Hill (2007) considers that deformation by buckling under horizontal stress is a function of excavation span, bed thickness and material stiffness properties. If this is the case, the current CMRR would capture the strength against buckling, but not the stress or load driving the buckling failure.

Horizontal stresses are typically more important for roof stability than vertical stresses. Most vertical stress is applied to the pillars, but the roof must take the majority of the horizontal stress. Mark (1991) found there to be significant variation in stress between and even within mines in the Western US and also that the horizontal and vertical stresses were approximately equal in the Western US. In the Eastern US, the horizontal stress was generally 2-3 times that in the vertical direction (Mark 1991). This somewhat contradicts the generalized statement made by Hill (2007) that the horizontal and vertical stresses in the US as a whole are generally the same and that this is the condition the CMRR is most applicable to. However, Hill (2007) seems to be comparing
the differential stress in the US to other countries, such as Australia saying stress is lower and more isotropic in the US. He therefore cautions that any empirical relationships using the CMRR and stress related parameters do not necessarily transfer well outside the US.

Horizontal stresses are rarely measured underground (Mark et al., 2001) and partly as a result of this, the ground stresses are also not well understood (Mark and Barczak, 2000). High biaxial horizontal stress is known to adversely affect roof stability (Mucho et al., 1995). Mucho et al. (1995) used overburden thickness as an analogue for horizontal stress, but in reality, actual stress levels vary based on region, direction of mining and surface topography (Molinda et al., 2000). Mark and Barczak (2000) also identified two other factors that determine the degree to which horizontal stress will affect ground control. The first factor was roof type. Weak roof is more likely to be damaged than strong roof, and the presence of fine laminations or bedding increased the susceptibility of the roof to horizontal stress induced damage (Mark and Barczak, 2000). The second factor was surface topography, in particular stream valleys which can concentrate horizontal stresses as well as reorient it away from the regional direction. With respect to the CMRR, the roof strength and discontinuities are captured, but not the topographic relief above the mine.

Stress measurements are too expensive to be used routinely by most mines (Mark and Barczak, 2000). Procedures such as mapping features like roof ‘gutters’ underground have been developed to estimate the orientation of the maximum principal stress (Mucho and Mark, 1994), however this is an inexact method. Mining can also reorient the original in-situ stresses. Similarly, when the roof deforms or fails, the
horizontal stress becomes concentrated where the roof is still intact (Mark and Barczak, 2000).

A roof fall in a geologically complex mine in China was investigated by Wang et al. (2018) to determine the main contributing factors. They conducted a series of field tests, studying the geological structures, in-situ stress and roof to floor convergence, among other parameters. The driving force of the roof fall was found to be high levels of horizontal stress and fault slip, induced by mining activities. The faults themselves (likely caused by compressive stress) were considered to be a major source of horizontal stress, exerting a greater force on the surrounding rock compared to where there were no faults (Wang et al., 2018). Griffith et al. (2014) also note that moisture degradation can increase where there is high horizontal stress, which can cause damage and increase permeability in the roof. Geologic ‘defects’ such as faults and their associated instability can be further made worse by the secondary effects of high horizontal stresses.

A number of roof fall types, such as stackrock delamination, cutter roof and spalling are all generally attributed to high horizontal stresses, with the normal stress acting parallel to the roof strata (Griffith et al., 2014). Stackrock is a sequence of mine roof rock made up of interbedded sandstone and shale layers. Cutter roof failure occurs due to the oblique orientation of the major horizontal stress in relation to roadway advance direction. Spalling refers to pieces of rock breaking off the face due to internal stresses. While the CMRR would record the discontinuities in the stackrock sequence, the likelihood of delamination would not be clear from the CMRR system alone as it
does not account for horizontal stresses. Cutter roof failure and spalling therefore cannot be predicted by the CMRR as it does not measure stress related parameters.

2.11.2 Intersections

It should first be noted that it is excavation span which affects roof stability, however intersections are the most common example of an area with a larger than normal span. The excavation span is not explicitly included in the CMRR, nor is the binary record of whether the data is collected at an intersection or not. Still, it seems that it may be included indirectly; if the roof is weakened by the increased load, this effect would be captured by the CMRR. Mark and Chase (1994) found a strong correlation between CMRR and entry width as did Molinda et al. (2000) who examined the relationship between the roof fall rate, intersection span and the CMRR.

The design of stable roof spans in underground mining is generally done using empirically based techniques. These may be supplemented by analytical or computational methods (Esterhuizen et al. 2011).

Rock load increases in proportion to the cube of the intersection span (Pappas et al., 2000), so even a small increase in span can significantly increase the rock load. In the cases examined by Pappas et al. (2000), they found widening the span from 18-20 ft caused the rock load to increase by 37%. Molinda et al. (1998) also conducted a study on intersection width at a mine in Pennsylvania and found 83% of roof falls occurred in 13% of the intersections where the span exceeded 70 ft. Mark and Barczak (2000) agree saying that approximately 70% of roof falls occur in intersections and are 8-10 times more likely to occur in an intersection than an equivalent length of entry.
Approximately 30% of fatal injuries in underground coal mines occur at intersections. 68% of these were found to occur at 4-way intersections; the other 32% occurred at 3-way intersections (Abbasi, 2010). This increased number at 4-way intersections is likely due to the greater effective roof span. Further studies on roof fall (causing fatal and non-fatal injuries) in the US by Spearing and Mueller (2008) and Chugh and Kollipara (2009) found that 70-75% of recorded roof falls in the US occurred at intersections.

The larger exposed roof span at intersections makes them more vulnerable to failure due to the associated stress concentrations. Sinha and Chugh (2018) studied critical failure strains in tension, compression and shear and through a case study, they demonstrated that modifying the opening orientation and installing reinforcement at critical locations can help improve the overall stability at intersections. Intersections are 3-dimensional, so early closed-form solutions using plate theory for stress distribution around an intersection were overly simplistic (Sinha and Chugh, 2018). Since the development of numerical modelling techniques however, researchers have begun to develop a better understanding of the stress conditions under realistic conditions like nonhomogeneous rockmass behavior and the sequential development of intersection. However, limited research has been done on designing roof support plans to improve intersection stability (Sinha and Chugh, 2018).

Gale et al. (2004) examined ground behavior and stress redistribution in a coal mine entry with high horizontal stress concentration. They modelled the specific situation of incrementally increasing far-field stresses at the mine using FLAC3D, relying heavily on roof extensometer data to estimate displacement at the roof line. Their approach was however aimed at ‘normal’, unfaulted and non-structurally complex
ground and assumed linear elastic behavior of the roof. Sinha and Chugh (2018) argue that using the approach by Gale et al (2004) where there is inelastic material behavior may not give a realistic estimate of failure and/or support requirements. They identify a need for better understanding of stress and strain distributions around intersections.

Another study by La Pointe and Clark (2015) also considers that roof at intersections is nearly twice as likely to fail as in entries due to enlarged roof spans and the re-distribution of stress. However, they note that the relative stability of intersections in a mine varies. In their work, they developed a model to assess the stability at intersections of entries and cross cuts, using an underground longwall mine as a case study. The statistical analyses they conducted were based on observational data on tension crack development, seepage, intersection geometry, mining practices, geology and geomechanical factors. They studied 783 intersections. The results of LaPointe and Clark’s (2015) analyses showed several factors impacted intersection stability:

- Overburden thickness
- Initial opening area
- Sulfur content
- Intersection type
- Gob distance
- Presence or proximity to a certain sandstone

La Pointe and Clark (2015) also found that the total intersection area and the supported intersection area showed no statistically significant relation to stability. They found tension cracks to occur preferentially where the sulfur content was low and suggested it related to mechanical differences in the roof rock. Their final conclusion
was that 2 and 3-way intersections presented fewer tension cracks than expected but 4-way intersections showed more. This implies that 4-way intersections are significantly weaker, agreeing with Abbasi (2010), as discussed above.

A high frequency of bedding planes can also be problematic at intersections; if the span of the opening is too wide the bed deflection can result in tensile cracking near the center of the opening or crushing along the edge. This excessive downward deflection can also result in the loosening of rock blocks and even total roof collapse (Esterhuizen et al., 2011). This seems to be particularly problematic when mining bedded stone deposits, but it seems likely to apply to all bedded deposits.

2.11.3 Faulting

Slickensides are failure surfaces with a history of shear movement. They occur most commonly in clay rich rocks: as grains coarsen, their ability to resist shear increases. Slickensides can form early in a rock’s diagenesis due to differential compaction or much later as a result of high confinement or slippage from tectonic or local faulting. Regardless of how they formed, slickensides are present in mine roof before mining; they are not caused by mining induced stresses (Molinda, 2003). Slickensides formed by differential compaction are commonly found beneath paleochannel margins, and where stiffer rocks abut softer ones (Molinda, 2003).

Slickensides are known to cause potentially hazardous ground control conditions in underground coal mines and seriously weaken mine roof (Molinda, 2003, Phillipson, 2003). Indeed, investigations by MSHA have shown that many slickensides essentially act as bedding plane faults (Molinda, 2003). Those that act this way tend to have been
formed in response to regional tectonic stresses (Phillipson, 2003). Between 1999 and 2003, MSHA found that many roof falls (often associated with fatalities) were due to rock separation along a slickensided surface. Many of the falls in question were in bolted areas, but the slickensides had gone unnoticed or their effect was underestimated. Slickensides are already accounted for by the CMRR, however, implying that such instabilities could have been captured by the CMRR at least in part.

Phillipson (2003) observes that the common perception of slickensides in the coal industry is that they are ‘randomly oriented, sporadically developed, localized discontinuities’. It is also assumed that they are ubiquitous to coal measure rocks. However, Phillipson (2003) argues that slickensides tend to occur in ‘predictable zones of geologic weakness’. They also tend to have a relatively consistent orientation within mines with their trend being parallel with the strike of local and regional features such as folds and large-scale faults. Slickensides that formed due to regional tectonic stresses can be associated with local structural geologic features and therefore mapped accordingly. Documenting such zones of weakness can therefore be relatively straightforward, yet can provide miners with a greater awareness of potentially dangerous conditions. It seems that slickensides are therefore a worthwhile inclusion in the CMRR.

Large faults, on a mine wide scale or similar, are not accounted for by the CMRR. Mark and Molinda (2005) explicitly state this, with the justification that such features “usually require specially designed support systems”. There do not seem to be any published examples of coal mine case studies with a major through-going discontinuity such as a fault, but it makes sense that a different approach to support
could be required across the entire mine. It could be that an adjustment factor could be added to the CMRR should there be a large-scale discontinuity, although significantly more information and data would be required to evaluate if this would be a worthwhile option.

### 2.11.4 Depth of Cover

The depth of cover matters in so far as how it affects the stress field. Currently depth of cover is not included in the CMRR.

Molinda et al. (2000) suggest that depth of cover is an analog for horizontal stress level. The overburden also contributes to the initial (unperturbed) vertical stress along with the specific weight of the overburden material. Molinda et al. (2000) found there to be a statistically significant correlation between CMRR and overburden. Their results implied that stronger roof was encountered as the overburden increased. Molinda et al. (2000) also investigated the relationship between CMRR, depth of cover and roof fall rate; they found 16 of the 22 cases of zero roof fall evaluated to be correctly classified by the CMRR and depth of cover and 19 of the 23 cases of high roof fall rate to be correctly classified. The overall correct classification rate using the CMRR and depth of cover was 77%.

The idea that stronger roof and by implication, more stable roof is encountered at greater depths seems to contradict what was observed at Mine B. However, it could be that CMRR is simply staying constant and the stress is increasing. In such cases, depth of cover may be taken as a proxy for horizontal stress.
2.11.5 Sandstone Channels

Large scale features such as sandstone channels are not included in the CMRR, although when the system was developed, Molinda and Mark (1994) explicitly acknowledged this. They suggested that a CMRR value could be developed for 'normal conditions' and another for areas where large scale features were present. This is potentially problematic when these large-scale features are pervasive throughout the mine and the most representative CMRR would be one which accounts for them, however.

According to Molinda and Mark (2010) there are four main ways sandstone (present as layers or paleo channels) can affect the roof. The first is where the sandstone is within reach of the roof bolt and provides a stable and competent roof. The second scenario is where the sandstone is slickensided or disturbed causing poor roof conditions. In the fluvial-deltaic environment of deposition common to coal measure rocks, it is not unusual to find evidence of channel scour and fill disrupting the continuity of the roof rock. According to Molinda (2003), paleochannel cutouts are associated with several adverse features. These include, but are not limited to faulting, crushed rock, slickensides, shearing, water, clay veins and slumping. During deposition, the peat swamp is scoured out by the channel which can cause slumping on the cut bank and cross bedding on the opposite side of the channel (Molinda, 2003). The slump surfaces can become slickensided by loading with the resulting disrupted blocks often falling between bolts. The instabilities from slip planes and faults associated with paleochannel deposits are worse when these structures are oriented parallel to face advance. In such
cases the mine roof can be segmented into cantilever beams (Ingram and Chase, 1987).

Third, if the sandstone is a groundwater aquifer, water can be introduced into the surrounding moisture sensitive shales, weakening them and greatly increasing the likelihood of roof falls. Lastly, stackrock can form on the margins of sandstone channels; this type of rock tends to concentrate horizontal stress, which can in turn cause roof damage (Mark and Molinda, 2010). This can occur due to differential compaction after burial of the paleochannel; the channel fill can point load the peat at its base and margins resulting in highly sheared and friable rocks at the paleochannel margins (Molinda, 2003).

Only in the first scenario is the presence of sandstone beneficial; for the other cases, sandstone, particularly as unpredictable and inconsistent channels, can significantly increase roof instability. It is therefore worth considering as a parameter over and above the CMRR which could also be indicative of roof instability. Recognition of common structures associated with paleochannels can also help identify their location and help with prediction, although this can be tricky, as many of these structures such as slickensides can a have a range of different causes.

2.11.6 Coal Seam Merge

Currently, the CMRR does not account for any effect of a rider coal seam above the main seam. It also does not account for the instability which could occur when the rider and main coal seams merge.
Rider coal beds are minor coalbeds up to approximately 4 ft thick which are the result of peat swamps reestablished on top of the main seam (Molinda, 2003). Rider coal is a loose term and may refer to a coal split at the top of the main seam too. The formation and preservation of rider coal depends on the depositional environment. In Southern Appalachia, the basin is deep and subsided faster meaning minor peat swamps were not preserved. In Northern Appalachia, subsidence occurred more slowly and rider coal is notably more common (Molinda, 2003).

Roof falls can tend to top out in the overlying rider seam, but this does not necessarily mean the roof beam first separated at the weak rider seam. It could also mean an arching discontinuity simply found a place to stop in the last weak roof member. Rider coals are generally considered to be weaker than the surrounding strata, as coal is usually weaker than shale and sandstone, although this is certainly not always the case. Rider coal can cause a variety of roof control problems, not least because of the challenges involved in mapping its location and thickness. If roof bolts are anchored in a rider coal seam, separation and roof fall can readily occur (Molinda, 2003). Ground Penetrating Radar has shown promise in mapping rider coals (Molinda, 2003), although regularly spaced boreholes (e.g. 20 ft) are a more popular approach which is considered sufficiently effective. Other geophysical methods may also be better suited to this application.

No case studies documenting a mine with ground control problems from a rider coal seam seem to have been published, although Mine B is an excellent example. Further information on Mine B and the ground control issues faced there may be found in Chapter 3.
2.11.7 Surface Topography

Surface topography is not included in the CMRR. Like depth of cover, it is likely to be related to the stress experienced by mine openings.

Moebs and Stateham (1984) reported that up to 90% of roof falls in underground mines in the Appalachian basin occurred beneath stream valleys. In 1992, Molinda mapped roof failures at five mines in Pennsylvania, finding 52% of them occurred directly beneath valley bottoms, whereas <10% occurred beneath hills. Molinda (1992) also found the chance of roof failure to be greater beneath broad valleys than steep V-sided ones, which could imply a gradual yet persistent change in slope to be more problematic than a sharp but less laterally extensive one. Griffith et al. (2014) suggest this increased failure rate below valley bottoms could be due to magnification of the horizontal compressive normal stress or from long term degradation of roof rocks due to fracture and fluid infiltration. Both of these relate back to the regional stress field being perturbed due to uneven topography.

Pan et al. (1994) found that under gravitational stress only, beneath irregular asymmetric valleys and ridges, there can be several local stress maxima and minima which could be sites of potential failure. Pan et al. (1995) found that the horizontal compressive stress at the bottom of the valley could be several times larger than the applied far-field tectonic stress. They also found that under a combination of gravitational and tectonic stresses, stress concentrations could be found on ridges. The topographically perturbed stress field strongly depends on the depth of the valley, the rock elastic properties and the strata orientation (Pan et al., 1995). The perturbed stress field below valleys can cause roof instability in shallow underground mines (Griffith et
al., 2014). However, it is not common for these stress perturbations to be quantitatively evaluated in the mine planning process.

Griffith et al. (2014) modelled the interaction between topography and tectonic stresses using the 3D boundary element method to evaluate the stress field induced at an individual mine. Specifically, they calculated the pre-mining topographically perturbed stress field at a case study mine in Ohio. Their calculated stress fields were then compared to mapped roof failure mechanisms throughout the mine. They found areas of high compressive stress to correspond to areas which commonly experienced cutter roof failure. This is a common failure mechanism in underground coal mines and is caused by the oblique orientation of the major horizontal stress with respect to roadway advance direction (Gao and Stead, 2013).

In shallow coal mines, stress related mechanisms are thought to be mainly controlled by the maximum horizontal compressive strength (Griffith et al., 2011). Topography only perturbs the stress field at the near surface and, especially where the depth and topographic height are of the same order of magnitude, the magnitude and orientation of the horizontal stress are generally considered to be essentially homogeneous throughout the mine (Griffith et al., 2011). Whether this is a valid approximation for all shallow coal mines is unclear.

Estimating the initial vertical stress in a coal seam is a trivial exercise in coal fields overlain by gently undulating topography but it is useful beneath steep or mountainous topography (Whyatt et al., 2011). The LaModel software (developed by Professor Keith Heasley of West Virginia University) is a discontinuity-displacement software which was designed to calculate the seam stress and displacement in an
underground mine, however Whyatt et al. (2011) feel it is less applicable to coal mines beneath “rugged” topography or in tectonically disturbed areas. Their study specifically looks at the estimation of in-situ stress in mountainous terrain in the context of the LaModel software. Whyatt et al., (2011) carried out a sensitivity study to evaluate how parameters such as overburden properties affected the estimated distribution of vertical stress. They found the spreading of stress in a coal seam away from ridgelines and into areas beneath valleys to be very dependent on the lamination thickness chosen; as the lamination thickness decreased, less and less spreading of stress was found to occur.

Development of an initial vertical stress estimate at a point underground depends on three main factors; the first is the depth and specific weight of the overlying strata. Secondly, an estimate of how undulations in the overlying strata could cause related undulations in the vertical stress field is required. The distortion of the vertical stress field by geological features such as faults, stratigraphy and coal seam dip is the third factor.

According to the World Stress Map (WSM, 2016), the coal fields of Utah, Colorado, Wyoming and New Mexico all fall within regions of extension of normal faulting where vertical stress is predicted to be greater than horizontal stress. This contradicts Mark (1991) and Hill (2007) who note that the k ratio (the ratio between the minor and major principle stress) is essentially 1 in the Western US. The Western regions are also seismically active in comparison with the East and in-situ stresses are considered much less predictable in seismically active coal fields (Mark and Gadde, 2010).
2.11.8 Rib and Floor Condition

The CMRR does not currently include the condition of the ribs and floor at the measurement location or their potential relationship with the stability of the roof. There does not seem to be any direct record of a situation where the mine roof was weak and the ribs and floor were too, although it seems very possible that the two could be related.

Rib sloughage and floor heave are commonly stress-induced and can cause a redistribution of stress around the opening, which in turn can cause significant ground control problems in the roof and elsewhere (Haramy and McDonnell, 1986).

2.12 Other Methods of Predicting Roof Instability in Underground Coal Mines

It is important to remember that the CMRR is just one method that has been proposed to help assess roof instability in underground coal mines. It is undoubtedly worth asking the question of whether there is a better method entirely.

In many mines, the operators have learned through trial and error how to mine the coal and support the roof. Given the significant variation in geological and geotechnical conditions in underground coal mines throughout the United States, it seems unlikely that this site-specific approach will change in the short term. In such cases, roof instability tends to occur when the conditions change (in particular geology) and the operator is unsure how to react or adapt (Molinda et al., 2000).

Another method similar to the CMRR was developed by Whittles et al. (2007). They developed a rockmass rating system for coal measure rockmasses that can be used to empirically predict their engineering properties. It is known that the strength and
deformation of a stratified rockmass varies with loading direction with respect to the orientation of the lamination planes, but there is no widely accepted method which accounts for this anisotropy. Most empirical equations in the literature proposed to estimate strength and deformation moduli assume the rockmass is isotropic. The classification system developed by Whittle et al. (2007) however, generates two numerical ratings corresponding to the different engineering properties in directions perpendicular and parallel to the stratification.

The use of geomechanical computational modelling in coal mine design means the strength and deformation parameters for rockmasses often need to be determined. The motivation for the work by Whittles et al. (2007) was therefore to develop a classification system specifically to predict the in-situ rockmass parameters of a stratified coal measure rockmass as required for input into numerical models. The geomechanical response of the rockmass surrounding underground coal mines is a combination of interactions between the rockmass, the high stress due to mining depth and the disturbances created by the mining itself (Whittles et al., 2007). According to Smith and Rosenbaum (1993), the parameters which are the most significant are those which affect the mode of failure due to the mining disturbance.

Romana (1995) states that any effective classification system has to include the different possible modes of failure. So, by default, a successful system should include and quantify the parameters which are most significant in these modes of failure. Whittles et al. (2007) identified five modes of failure associated with the roof strata in coal measure rock. These are:

- Buckling of roof beds due to horizontal stresses
• Self-weight sagging of roof beds
• Shear failure of roof beds
• Shear joints/parting plane failure
• Wedge/block failure

Whittles et al. (2007) went through a similar process in developing their Coal Mass Classification (CMC) system as Molinda and Mark (1994) did with the CMRR. Their first step was to identify the most significant parameters; in reviewing the literature, they found fifty different parameters which were used in other classification systems (not just those for coal mines). Next, they identified which parameters on this list had the greatest effect on the deformation of a stratified rockmass. To do this, they evaluated each parameters effect on the identified potential failure modes and ranked them on a scale of 1-6 with 1 being the most significant and 6 being the least. From this, they identified 10 mutually independent parameters as the most influential on strata deformation. Whittles et al. (2007) then divided these parameters into 6 main categories:

• UCS of the intact rock
• Bedding/lamination characteristics (spacing and strength)
• Joint properties (set number, orientation, strength, set spacing)
• Degree of fissility (shaleyness)
• Water flow
• Sensitivity to changes in moisture content

These parameters are very similar to those included in the CMRR.
A more field-based method for the prediction and identification of roof fall is the use of seismic and infrasound techniques. Zimmer and Sitar (2015) detected rockfalls on a historically active cliff using seismic methods. They were able to develop a triggering algorithm and criteria to distinguish rock fall from thousands of seismic triggers with the end result being a set of characteristic parameters indicative of rock fall which could be seismically detected. While their study was not in an underground coal mine, there could certainly be potential for technology transfer.

NIOSH has also developed the Roof Fall Risk Index (RFRI) to systematically identify roof fall hazards in underground stone mines (Iannachione et al., 2007). The RFRI focuses on the character and intensity of defects from mining, geologic and stress factors. The total rating is on a scale from 0 to 100, with 0 reflecting stable roof conditions and 100 a serious roof fall hazard. The RFRI requires roof fall hazards to be mapped and their spatial distribution within the mine evaluated. The aim of the RFRI was to use fewer time consuming measurements than are required in other rockmass classification systems, yet still evaluate roof stability over a large continuous area (Esterhuizen et al., 2007; Iannachione et al., 2007). This method has been used predominantly in stone mines. The extent of its use in coal mines is unclear, but it could prove a valuable method in combination with the CMRR for evaluating roof instability in underground coal mines.

Vaziri et al. (2018) used a GIS based approach to develop a geological-geotechnical risk assessment model to identify areas of high risk in underground coal mines. They estimated the CMRR and their other hazard/risk factors, then used an interaction matrix method to weight each of these factors, overlaying them to create a
final geohazard risk map. Comparing their results to previously mined out areas, they found generally good agreement.

Oraee et al. (2016) analyzed the structural data and the geometry and stability of wedge failure in underground mines, using software such as UnWedge to back analyze the potential cause of failure. Also collecting field data, but taking a different approach, Palei and Das (2008) suggested that the support safety factor could be a useful predictor of underground roof fall. They collected geotechnical data from 14 locations of roof fall incidents in an underground coal mine and estimated the mean value of the probabilistic support safety factor. They then conducted a sensitivity analysis to evaluate the effects of the input parameters on support safety factor and the likelihood of roof fall. Multi-variate regression was carried out for the data to correlate the contributing factors to the support safety factor. Their results showed that a higher CMRR (or RMR) correlated with a higher support factor.

In a more mathematical approach, Alejano et al. (2008) used different back analysis approaches to better understand the causes of a roof bed failure in an underground mine and to define the most significant parameters affecting the failure. Namely, these methods were the empirical stability graph method, analytical voussoir techniques and the numerical discrete element method. The empirical stability graph method calibrates hanging-wall or roof stability and is based on a large database representing unsupported stopes in Canadian mines.

Analytical voussoir techniques are based on the idea that persistent bedding in stratified rocks makes the rockmass highly anisotropic. Over an underground excavation, the only load on a detached stratum is its own weight. Treating this as a
beam allows stress distribution and flexural moments to be calculated. This theory assumes the roof is a series of equal width beams with pillars on either side. In reality, for tensile failure to occur, vertical discontinuities are required too, making the beam act as a voissoir beam. In this case, there is a compressive arch acting along the voussoir, withstanding the load and transmitting it to the abutments. To solve the voissour beam equations, at least one reasonable assumption is required and there have been several different solution methods proposed (Sofianos 1996; Diederichs and Kaiser, 1999). Using the voissoir beam theory, beam buckling, failure by crushing or spalling and shear failure at the abutments can be studied.

Discrete element method codes such as UDEC can also be used to solve the Voussoir beam problem numerically (Alejano et al., 2008). However, the geomechanical parameters of the rock and discontinuities input need to be realistic for the model to give worthwhile results. For this reason, Alejano et al. (2008) recommend a combination of analytical methods and numerical methods. Sherizadeh and Kulatilake (2016) also used the discrete element method to assess roof stability in a room and pillar coal mine in Pennsylvania. To accurately model the post-failure behavior of the immediate roof, the strain softening method was used. After conducting a sensitivity analysis on the rockmass strength properties, discontinuity properties and orientation and magnitude of the in-situ stresses, the distribution of post failure cohesion and accumulated plastic shear strain along with other parameters were used to assess roof stability. Sherizadeh and Kulatilake (2016) found bedding planes to be particularly influential in roof behavior in underground excavations and recommends that any modelling technique used for
this purpose needs to realistically simulate discontinuous rockmasses and the layered strata of underground mines.

Razani et al. (2013) used a fuzzy inference system to predict roof fall in underground mines. The fuzzy inference process maps a given input (e.g. roof strength) to an output (e.g. roof stability) using fuzzy logic. Fuzzy logic is such that the truth variables may be anywhere between 0 and 1 essentially covering the idea of partial truth – the truth can range anywhere from completely true to completely false. This allows truth to be a continuous variable. Comparing their results to artificial neural network and multivariate regression models, Razani et al. (2013) found the fuzzy logic method gave more accurate results. Rafiee and Azarfar (2018) also used fuzzy logic in their work. Their idea is that due to the uncertainty in the CMRR data collection, the final result may be inaccurate. To try and minimize this, they designed a fuzzy logic system to calculate the CMRR, using only the quantitative CMRR variables (UCS, joint spacing and persistence) as fuzzy inputs. Using their methods, Rafiee and Azarfar (2018) then added the fuzzy system output to the score obtained from the qualitative CMRR variables to get a final fuzzy CMRR value.

Although their study focused only on intersection stability and not the mine roof as a whole, LaPointe and Clark (2015) also looked at common data mining techniques such as multivariate linear regression, multinomial logistic regression, decision trees and probabilistic neural nets as possible methods to establish correlation between stability and the other variables they examined. They found decision trees and multinomial logistic regression to be the most successful and showed that several
factors impacted intersection stability, including overburden thickness, initial opening area and intersection type.

Another approach was the use of an artificial neural network (ANN) by Mahdevari et al. (2016). They monitored roof displacement at various locations along the mine roadway and used these as inputs to the ANN model. Further geotechnical parameters found through site investigations and laboratory tests were also added to the model as independent variables. The unknown non-linear relationship between the rock parameters and the roof stability were then estimated by the model. Mahdevari et al. (2016) found a high level of correlation between the predicted and measured roof displacement values concluding that their proposed ANN model is suitable for the prediction of roadway roof stability in underground mines.

These computational and mathematically based approaches are definitely valuable, however, the CMRR is arguably a more straightforward and easy to use tool in the field. If a mine is unlikely to use the CMRR as it stands, it seems more unlikely they would use a more intricate and complex approach such as that in one of the studies discussed above. It is also unclear whether the mathematical approaches are any more widely applicable among mines with varying geology and/or stress conditions than the CMRR.

Roof falls are commonly studied using a probabilistic or empirical approach (like the CMRR); it is difficult for deterministic rock mechanics models to fully explain why a given intersection collapses yet those adjacent remain stable (Molinda et al., 2000). The common empirical approach involves collecting a large volume of case history data and statistically analyzing it to find the most influential factors. As Molinda et al. (2000) write,
much can be learned from the observation of roof instability. The geology, geometry, timing and frequency of roof falls can usually give some indication of the cause of failure. Documentation of these variables on a mine-wide basis could make it possible to characterize the combination of factors which contribute to a high number of roof falls. In his summary of ‘topical areas of research needs in [mine] ground control’ Peng (2015) agrees with this. He states that most ground control failures are related to geology and understanding as much as possible about variation in the mine geology is particularly valuable.

It would therefore seem that as a field based, relatively easy to use method, the CMRR provides a good starting point for further research. However, in order for it to be applicable at a range of mines in varying geological and geotechnical conditions, there seems to be opportunity for improvement. Aside from the study by Taheri et al. (2017) who experimented with adding width of gallery and density of overburden rock to the CMRR, there do not seem to be any studies which attempt to make the CMRR system more comprehensive or applicable to Western US coal mines. If this is possible, it would prove a very useful tool in the prediction of roof fall in underground coal mines and could decrease the number of injuries and lost days even further. An improved understanding of what factors impact roof instability can lead to efficient, proactive mining practice, and this in turn will improve both stability and mining efficiency (Lapointe and Clark, 2015).
CHAPTER 3
CASE STUDY MINES

3.1 Introduction

Relationships have been developed with Mines A and B as part of a larger NIOSH-sponsored capacity-building project within which this research was conducted. These mines are used as case studies for this thesis research. A summary of the mine geology at each site is given below, with a focus on ground control.

3.2 Mine A

This mine is located in New Mexico and exploits a coal seam by underground longwall methods.

3.2.1 Regional Geological Setting

Mine A exists in a basin that is a structural depression formed during the Laramide orogeny (approximately 75 to 45 million years ago). On a larger scale, this orogeny was due to compressional forces from subduction which ultimately caused the uplift of the Rocky Mountains. The basin is approximately 140 miles wide and 200 miles long, with a maximum relief of about 10,000 ft and is a classic example of basin structural evolution that is penecontemporaneous (occurring immediately after deposition) with basin sedimentary deposition (Fassett, 2000). The tectonic evolution of the basin is extensive and complicated. The Precambrian host rock has experienced metamorphism, deformation, and erosion before being buried under Phanerozoic rocks.
Tectonism began in the late Paleozoic age and continued through Mesozoic times, although the majority of the structural elements present in the basin formed during the Laramide orogeny (Craigg, 2001). The structural elements of the basin are generally split into three categories: large elongated domal uplifts, low marginal platforms and sharp monoclines. The Nacimiento uplift is a particularly notable feature of the basin, representing the SW limits of the Rocky Mountains (Craigg, 2001). It is a north trending mountain block approximately 50 miles long comprising a block of Precambrian rock that has been thrust westwards. Its Western flank consists of two separate major faults and is considered to be structurally complex (Woodward, 1987).

The formation containing the coal beds currently under exploitation is of Upper Cretaceous age and the area of the basin within the formation is approximately 6,500 mi² (Fassett, 2000). The basin as a whole contains about 6000 ft thick late Cretaceous rock; these rocks are a series of interbedded marine and non-marine strata deposited during a series of transgression and regression cycles. The final regression is thought to have occurred around 75 Ma at which time a sandstone unit was deposited (Fassett, 2000). Fassett (2000) interpreted the sandstone to be deposited dynamically as the shoreline regressed across the basin area; as regression occurred, sediment was deposited simultaneously in the continental, shoreface and marine environments. The coal beds were conformably deposited on top of this sandstone unit in backshore swamps. Another hypothesis for the deposition is that presented by Ayers and Ambrose (1990), suggesting differential tectonism controlled the regression of the sandstone and the accumulation of the coal beds. This is worth noting as it could explain (at least partially) the high incidence of localized faulting at Mine A.
Volcanism is thought to have occurred during the deposition of the coal beds, as well as uplift. After deposition of rocks in the Late Cretaceous period, the basin was tilted to the NW before a basin-wide erosion cycle occurred (Fassett and Hinds, 1971). Subsequent deposition was accompanied by uplift and erosion of the formation hosting the coal beds. Volcanic activity in the North of the basin led to further deposition, before renewed uplift and associated erosion occurred again (Fassett and Hinds, 1971). While some of the details within the depositional and structural history are still debated, it is clear is that the entire basin has seen several uplift and erosive events which have led to present day structural complexity. The formation that hosts the coal beds being mined comprises mudstones, siltstones, sandstones, carbonaceous mudstones and coals.

3.2.2 Local Geologic Setting

There are multiple coal seams within the host formation. The coal seams are numbered in order of superposition. The coal seam exploited by Mine A will be called Coal Seam H for the purposes of this research. This seam dips very shallowly (1-3°) towards the E-NE and is intersected by a series of ash bands. Coal seam H consists of five coal benches and four volcanic ash band partings (also labeled sequentially based on superposition) (Burkhard and Herth, 2016). Several paleo sand channels are present in the layers overlying Coal Seam H, and the mine engineers have noted that the channels influence the ground control. This stratigraphy at Mine A is represented schematically in Figure 3.1.
Figure 3.1: An idealized cross section of Coal Seam H. Coal benches, volcanic ash bands sandstone channels and roof and floor mudstone are shown. The red lines are schematic representations of faults that occur within the mine area. This figure is not to scale, however the approximate height of the section is 100 ft. (Burkhard and Herth, 2016).

The immediate roof and floor at Mine A are composed of mudstones, carbonaceous mudstones and siltstones (Pile et al., 2003); the mudstone is particularly susceptible to being wetted and will easily deteriorate in the presence of water. This is considered to be due to high swelling clay content and weak cementation (Burkhard and Herth, 2016). There is also localized faulting present in the mine. This generally presents as slickensides or as faults of length up to 2 ft. The maximum fault
displacement observed was approximately 1 ft. The slickensides were observed to be concentrated in certain areas throughout the mine and less frequent in others.

The near-seam sandstones (see Figure 3.1) are generally fine to medium grained, poorly graded and contain fines in varying amounts. This is based on core logging conducted in the strata 40 ft above to 40 ft below Coal seam H (Burkhard and Herth, 2016).

A series of laboratory and field tests were carried out on intact rock samples from the coal and adjacent strata at Mine A. However, they were conducted from limited data and notable variation in the results was encountered (Burkhard, 2016). The results from these tests are shown in Table 3.1. Burkhard and Herth (2016) note that the true mean strengths for the weaker mudstones and siltstones may be lower than indicated by the laboratory tests, primarily due to sampling difficulties. Burkhard and Herth (2016) also used geotechnical core log data to classify the different lithologies using the RMR and Q systems. The results of their analysis are presented in Table 3.2.

Table 3.1: Results of intact rock testing at Mine A (Burkhard and Herth, 2016).

<table>
<thead>
<tr>
<th>Lithology</th>
<th>Unconfined Compressive Strength (psi)</th>
<th>Young's Modulus (x10^6 psi)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Coal</td>
<td>790-1660</td>
<td>0.28-0.41</td>
</tr>
<tr>
<td>Mudstones</td>
<td>190-800</td>
<td>0.01-0.28</td>
</tr>
<tr>
<td>Siltstones</td>
<td>740-7420</td>
<td>0.13-1.65</td>
</tr>
<tr>
<td>Sandstones</td>
<td>4340-7290</td>
<td>0.94-1.73</td>
</tr>
</tbody>
</table>
Table 3.2: Results from classifying the lithologies present at Mine A using the Q and RMR systems (Burkhard and Herth, 2016).

<table>
<thead>
<tr>
<th>Lithology</th>
<th>Q Geometric Mean</th>
<th>RMR Mean</th>
</tr>
</thead>
<tbody>
<tr>
<td>Coal</td>
<td>13.2-23.7</td>
<td>50-61</td>
</tr>
<tr>
<td>Mudstones</td>
<td>5.5-9.4</td>
<td>34-52</td>
</tr>
<tr>
<td>Siltstones</td>
<td>11.3-18.2</td>
<td>45-72</td>
</tr>
<tr>
<td>Sandstones</td>
<td>23.4-51.7</td>
<td>58-74</td>
</tr>
</tbody>
</table>

3.2.3 Ground Control

The main influences on ground control at Mine A are considered to be the geological structure and the corresponding properties; indeed, it is a combination of many individual geological features (rather than one single condition) which seems to produce the most adverse conditions. Regional scale features are thought to have little direct influence on ground control at the mine (Burkhard and Herth 2016); rather, the local physical, structural and depositional features of the above seam strata have the greatest effect.

The following parameters are considered by the ground control engineers to be influential on ground control at Mine A:

Mining Horizon: When mining at depths >800 ft, a lower mining horizon is used to allow some coal to remain in the roof. Roof coal improves the roof stability and helps lower the risk of roof instability.

Depth of Cover: Mining depth ranges from 450 ft to 1100 ft. Clear correlation between depth of cover and ground control issues has not been shown, but it is thought that when combined with other unfavorable conditions, the increased stress may exacerbate existing ground control problems (Burkhard and Herth, 2016).
Surface Topography: The surface topography at Mine A consists of rolling hills that transition to cliffs approximately 200 ft high. Where the topography changes, an increase in ground control issues has been observed in the form of increased rib damage and signs of roof instability.

Ash Bands: Four ash bands are present in Coal Seam H and form distinct planes of weakness. Minor deformation of these bands is common and is rarely considered to be a ground control issue. Roof layer delamination however, has been observed to occur along ash bands 6-24 inches within the roof.

Faults and Slickensides: The coal seam and the layers immediately above and below are densely faulted. The faults and slickensides contribute directly to roof fall (Burkhard and Herth, 2016). Where faulted roof is encountered, the most notable result is roof fall. The roof falls leave a domed area in the roof, the size and depth of which is controlled by the size and orientations of the faults. If there is no immediate roof fall upon mining, faults and slickensides tend to cause problems later. As stress redistributes during and after mining, these discontinuities provide planes of weakness for displacement to occur along as well as potential pathways for water. Fault displacements may be in excess of 9-10 ft, but most range from a few inches to 2 ft. The highest frequency of faults is in the mudstone which immediately overlies Coal Seam H (Burkard, 2016).

Sandstone Channels: Two sand channels are present in the mine roof. The first (Channel 1) is closest to the coal seam, ranging from direct contact with the top of the seam to >20 ft above the seam. The second (Channel 2) is stratigraphically above the first channel and ranges from 40-120 ft above the seam (Burkhard and Herth, 2016).
The closer Channel 1 is to the immediate roof, the greater its influence on ground control. During the period of coal formation, Channel 1 was a major drainage channel which scoured out the coal seam to the NW of the mine creating a 'splay channel' over the seam (Burkhard and Herth, 2016). These channels are associated with widespread compactional faulting and trapped water. Channel 1 is considered to be the main source of water in the roof and to have the greatest influence on ground control. A schematic depiction of the location of Channel 1 above Mine A is shown in Figure 3.2. Burkhard (2016) found there to be a correlation between the location of the channel margins and areas of localized faulting. Faulting is not restricted to the channel margins, however when they are correlated, it is thought the channel margins were a driving force in the formation of the faults. The scour caused by Channel 1 controls the thickness of coal which can feasibly be left in the roof. In areas with significant scour, the bolted interval is commonly without any ‘protective’ coal below the weak mudstone. Areas such as this tend to cause roof control problems at a later date. Conversely, however, this scour can also be deep enough that the bolted interval is almost entirely in sandstone (the strongest stratum in the mine) which is beneficial and detrimental at the same time, as it also introduces water into the roof.

Water: Roof water is one of the greatest concerns with respect to ground control at Mine A. Roof water comes from the SS1 channel which has low transmissivity and permeability values (Burkhard and Herth, 2016). The water is therefore considered to be trapped or ‘perched’. Water in the immediate roof causes the loss of mechanical strength of the roof, particularly where the immediate roof is mudstone. The mudstones at Mine A are inherently weak and when wetted, their load-bearing capacity is notably
decreased, making the roof significantly more unstable. Water inflow can cause an area of mine roof with no previous visible indication of instability to require standing support within a few days simply due to the degradation of the roof mudstone (Burkhard, 2016).

Figure 3.2: Schematic Illustration of the Channel 1 Sandstone in the roof of Mine A. The panel names have been removed to preserve the anonymity of the mine.

Geomechanical Properties: The immediate roof at Mine A is inherently weak (Burkhard, 2016). The predominant lithologies in the bolted roof are mudstones; both carbonaceous and silt rich. As seen in Table 1, the relatively higher strength of the coal is the reason a remnant thickness is usually left in the roof during mining to help strengthen the bolted interval. This coal layer also protects the mudstone from
deteriorating significantly in the presence of roof water. The low UCS of the immediate roof mudstones (1000-2000 psi), the presence of closely spaced bedding planes, slickensides and other discontinuities led Pile et al. (2003) to estimate the mine-wide CMRR as 30 to 35, although no systematically derived value seems to have been established by the mine itself.

Pile et al. (2003) also note that the in-situ stress is relatively benign. Determination of the in-situ stress orientations and ratios were carried out during the initial underground pilot project at the mine; however, Burkhard (2016) reports it cannot be stated with any certainty the extent to which in-situ stresses influence ground control at the mine. He notes that the relative orientations of gate road and cross cut orientations, with respect to the principal stress, influence immediate roof stability but the influence cannot be quantified accurately at present. Pile et al. (2003) however, noted that “no clear indication of roof damage caused by horizontal stress have been observed in the underground workings”. It is worth noting however, that the horizontal stress may not be high enough to cause new fractures to form. In this case the horizontal stress could still contribute to slip along existing fractures presenting in a similar way to gravity driven failures. Horizontal stress would therefore still contribute to roof damage although there would likely be “no clear indication” that the damage was caused by horizontal stress.
3.3 Mine B

This mine is located in Montana and exploits bituminous coal from a 10 ft thick seam using underground longwall methods. The geology is considered to be relatively uniform but there are two unique geological features: low overburden and a distinct pattern of roof joints (Hanson et al., 2015).

3.3.1 Regional Geologic Setting

Mine B lies in an approximately elliptical basin 50 miles long and 30 miles wide, with the long axis pointing northeast (Woolsey et al., 1917). The mountains hosting Mine B are east of the larger Rocky Mountain chain. In comparison to surrounding mountain ranges, these mountains are relatively low lying with a maximum elevation of approximately 4,700 ft. The maximum topographic relief in the area is approximately 1800 ft. Woolsey et al. (1917) noted the topography of the region to be “peculiar” and notably different from that of the surrounding country. The coal beds in the region are hosted by alternating beds of resistant sandstone and clay shale which lie close to horizontal throughout most of the basin (Woolsey et al., 1917). Where these types of rocks are eroded in a semi-arid environment, the result is a series of flat plateaus preserved by the sandstone beds and steep, sharply cut valleys where rivers have eroded the sandstone to the weaker shale below. Outside the area of the coal field, there seems to be less differential resistance to erosion within the outcropping rocks and the topography is generally rolling prairie (Woolsey et al., 1917).

The primary coal-bearing formation was deposited in the Eocene epoch of the Tertiary period. This formation is present across the entire coal field with a thickness of
approximately 1650 ft (Woolsey et al., 1917). The rocks in this formation are generally massive sandstone alternating with shale/clay and coal. The sandstone beds can range from a few inches to greater than 50 ft in thickness and have a soft, loose texture with very little to no trace of stratification. The sandstone weathers into smooth, rounded faces and has been observed to form near vertical cliffs approximately 50 ft high. However, despite the local variation in sandstone thickness over short distances, the sandstone beds are very consistent and can be traced across the entirety of the region (Woolsey et al., 1917). The interbedded shale is clay-like and soft; in many places it grades into shaley sandstone. The abundant fossils and freshwater shells present in the formation as well as multiple coal beds imply that these beds were deposited in fresh or almost fresh water with significant vegetation on the surrounding shores.

The geologic structure of the region is relatively straightforward (Woolsey et al., 1917) with a broad, gentle synclinal basin comprising almost the entire coal field. Towards the east and south, the syncline flattens and the bedding becomes almost horizontal in the limbs of the fold. The dips of the beds within the syncline are steepest on the north side of the fold; south of the axis of the syncline, the dip is irregular but trends towards the northeast at low angles (1-4°). The coal thickness is essentially constant throughout the field; where it thins out, the edge of a new lens generally begins a short distance away at the same horizon or within 5 10 ft (Woolsey et al., 1917).

3.3.2. Local Geological Setting

At Mine B there is a primary coal seam and a rider seam. They are separated by an interval of interburden which is so small in some places that the coal can be mined
as a single bed, but in other locations the distance between the beds can be as much as 25 ft. Where the rider seam is within 12 ft of the immediate roof it may be more unstable and additional support may be required (Hanson et al., 2015). The coal is high grade, sub-bituminous and generally uniform in character (Woolsey et al., 1917).

At Mine B, the primary coal seam is overlain by up to 800 ft of Eocene sedimentary rocks. The overlying formation is made up of sandstone, siltstone, claystone and 26 other minor coal seams which are generally thin. The roof shales are easily softened (weakened) with the addition of water.

The roof sandstones occur in two facies: braided alluvial channel and river flood overbank deposits. Both are lenticular; the channel deposits are continuous up to several square miles, but the overbank deposits tend to occur locally and are generally thinner. The channel sandstones tend to outcrop on the plateaus of higher relief and may be anywhere between a few feet to 60 ft thick. They are fine to medium grained with channel and bar structures and large-scale cross bedding (Maleki and Gulliver, 1988). There are some minor silt or clay partings and Maleki and Gulliver (1988) observed the cores to break as easily across the laminations as along them. The overbank sandstones are interspersed with siltstones and shales. These siltstones and shales tend to be fan deposits with upward coarsening, gradational sequences. Ripple laminations are common as is fine carbonaceous material on the laminations, along which core breaks easily (Maleki and Gulliver, 1988).

A generalized, schematic lithological log of the mine strata is shown in Figure 3.3. The average UCS values for each unit are also shown in Figure 3.3. Maleki and Gulliver (1988) found the roof rock quality and strength to be ‘ideal for excellent
stability’, as it is predominantly sandstone and siltstone with an RQD of 96 and a UCS of 3770 psi. They conclude that this massive roof would require very little support for cover depths of less than 800 ft in the region where Mine B exists. The exceptions to this are where the immediate roof is thin carbonaceous shale or where the rider coal seam is within 10 ft of the immediate roof. The shales present in the overburden and near seam rock at Mine B were found to be fissile, weak and plastic (Maleki and Gulliver, 1988). Their strength was found to increase as the silt content increased leading to a range in UCS values from 2700 psi to 8750 psi for the shales and shale/siltstones.

The sandstones are weak but massive with an RQD of between 70 and 100 and an average UCS value of 4000 psi. Indeed, the strength and deformation properties for the different lithologies are all very similar, around 4000 psi and relatively soft (Maleki and Gulliver, 1988). Perhaps the main exception is that the sandstone in the immediate roof is significantly stronger than the sandstone unit higher up in the strata, (see Figure 10) with average UCS values of 4800 and 2300 psi, respectively. The coal in the main seam was found to be weaker than the immediate roof (Maleki and Gulliver, 1988) with a UCS of approximately 2000 psi. Leaving coal in the roof is therefore not considered necessary as a stability precaution.

No evidence of faulting at Mine B was found by Maleki and Gulliver (1988) or in the MSHA linament study of the area (Ferriter, 1988). Maleki and Gulliver (1988) hypothesize it would be unlikely for faults with throw greater than a foot to occur in the area. If there were any faults in the area, it is thought they would be related to the
Maleki and Gulliver (1988) tested 26 samples of the immediate roof and floor for elastic properties (Young’s modulus and Poisson’s Ratio). They found the elastic properties for the shales, sandstones and siltstones to not be significantly different and depositional environment rather than to tectonism, and therefore that the faults would be local and not likely significant surfaces of weakness (Maleki and Gulliver, 1988).
that all the material was relatively soft and deformable. The results of this testing are summarized in Table 3.3.

Table 3.3: Elastic Properties for Immediate Roof at Mine B (Maleki and Gulliver, 1988).

<table>
<thead>
<tr>
<th>Rock Type</th>
<th>Young’s Modulus (GPa)</th>
<th>Poisson’s Ratio</th>
</tr>
</thead>
<tbody>
<tr>
<td>Shale</td>
<td>3.99</td>
<td>0.12</td>
</tr>
<tr>
<td>Sandstone/Siltstone</td>
<td>3.79</td>
<td>0.16</td>
</tr>
</tbody>
</table>

3.3.3 Ground Control

Based on several years of observation and experience of the mine specific conditions, the mine engineers at Mine B report that the main influences on ground control are depth of cover, slope grade of topography and the merge of the main and rider coal seams. These are discussed briefly below. The mine engineers have developed a support plan based on these criteria or combinations thereof. Given the relatively consistent nature of the mine stratigraphy, the presence of geological features that are influential at Mine A such as ash bands and weak mudstone seem to be insignificant when assessing roof instability at Mine B.

Depth of Cover: The mine engineers consider a depth of cover of greater than 600 ft to be the ‘cut off’ for installing increased support. This is based on their observation and experience.

Surface Topography Grade: For a topographic slope angle approximately greater than 33° (a 3:1 slope), the mine engineers will install additional support. This is again based on observation and experience. Since the topography above the mine contains some steep cliffs, increased support due to steeper topography is not uncommon.
Coal Seam Merge: Where the rider seam is within 4-12 ft of the roof and therefore the top of the main seam, increased support is used. Once the rider seam is within a couple of feet of the main seam, it is not considered a problem as both coal seams are mined out together. It is also not considered problematic when it is more than 12 ft above the immediate roof as this is sufficiently far above the bolted interval. In comparison to Mine A, Mine B is much less complex geologically, meaning the parameters thought to influence ground control are much fewer. This also meant there was less need for the mine to investigate or calculate further geotechnical parameters such as the Q or RMR values.
CHAPTER 4

RESEARCH METHODS

4.1 Introduction

The following steps were used as guidelines to address the problem statement and hypothesis outlined in Chapter 1.1.

4.2 Literature Review

This included gathering information on the development of the CMRR, its strengths and weaknesses and examples of its use, both within the US and internationally. An examination of other methods of quantifying coal mine roof instability and a discussion on why the CMRR may be less widely used in the Western US was carried out. Parameters also thought to be predictive of roof instability, but not included in the CMRR (e.g. depth of cover) were also examined. This step also included background reading on both mines to be used as case studies and an overview of the geological conditions at each, as well as any roof instability or ground control issues faced and what was done to address them.

4.3 Initial Observations

Mine tours and meter-scale mapping of the mine roof in areas of interest were carried out. Interviews with the ground control engineers were also conducted at both mines. This was enabled the researchers to build relationships with these mines and
also to make initial observations of the similarities and differences between them. These activities were important preparatory steps for forming a hypothesis to be tested.

From the literature review and initial mine visits, the similarities and differences between the mines were examined. It was found that the CMRR was used at neither mine to predict roof instability; at Mine B it was used for the Analysis of Longwall Pillar Stability (ALPS). Notably at Mine B, the laterally consistent and relatively stable geology did not seem to correlate with roof stability; there were still areas of stable and unstable roof despite the consistent geological conditions. The CMRR is focused on the character of the units of the immediate roof; if the geology is the same, the CMRR values throughout the mines should be the same, thus implying the roof stability throughout the mine is the same. These observations therefore suggested there were other factors over and above those included in the CMRR which could also predict roof instability. At Mine A, the geology was more variable laterally, and there were certain large-scale features – in particular the paleo sandstone channels, which were considered to have a significant influence on ground control by the mine engineers. Such large-scale features are not currently included in the CMRR.

4.4 Formation of a Hypothesis

The above observations led to the formation of the following problem statement and hypothesis which could be tested:

Problem Statement: What geological and geotechnical factors in addition to the existing CMRR criteria could also be predictive of roof instability at these two case study mines?
Hypothesis: The parameters currently considered by the CMRR do not fully explain roof instability. Where the input parameters currently considered by the CMRR remain relatively constant i.e. the immediate roof geology does not change laterally, there are other factors that are correlated with roof instability in underground coal mines and which could potentially be included in the CMRR.

4.5 Development of a Fieldwork Plan

At each site, data pertaining to the parameters considered to be potentially predictive of roof instability but not included in the CMRR were also recorded. These data included depth of cover, surface topography grade and the presence of sandstone channels. A full list of the data collected at each site is summarized as follows:

- CMRR values (including component values)
- Roof stability (stable or unstable)
- Depth of cover
- Slope grade of surface topography
- Topographic curvature
- Merge of coal seams present? Distance between seams.
- Faulting present? Localized or not?
- Sandstone channels present?
- Located at an intersection?
- Rib and floor condition?

The number of measurement sites (30 at each mine) was determined such that there was enough time to collect the data in the two days spent at each mine and be
sufficiently thorough in doing so. The measurement locations were selected randomly, representing all areas of the mine which were accessible. At Mine A, several data points were taken behind the start line (see Figure 4.1) as this was where the paleo sandstone channel is present in the immediate roof. At Mine B, the sites were selected such that a representative selection of locations under deep cover, steep slopes and within 12 ft of the coal seam merge were included. The data collection locations are shown on the mine maps in Figures 4.1 and 4.2.

Collecting these data allowed the hypothesis to be addressed by evaluating the correlation between the roof stability and CMRR or the non-CMRR parameters which could also be indicative of roof instability in underground coal mines. It was initially considered that, where possible, the CMRR data collection locations would be selected based on the existence of available core logs and/or samples. This would theoretically improve the accuracy of the CMRR measurements and the ability to obtain information not visible by mapping, such as the number of bolted intervals in the roof or the presence of a strong bed within the bolted interval. At Mine A there were 8 locations where core had been drilled and which were still accessible, although the core itself was only available for 2 of these sites. At Mine B there were no accessible sites for which core and logs were available. It was therefore decided to not pursue the core log CMRR method. In order to keep the measurements as consistent as possible, all data were based on observations collected underground.
Figure 4.1: Map of Mine A. Red stars denote the data collection locations. The isolines represent depth of cover in feet.
Figure 4.2: Map of Mine B. Red stars denote the data collection locations.
4.6 Fieldwork/Data Collection

The above fieldwork plan was followed and the data collected at both mines at each of the sites shown in Figures 4.1 and 4.2. In order to test the above hypothesis, the data required to calculate the CMRR (e.g. UCS estimate, joint persistence etc.) were collected at 30 sites at each mine (60 total). Detailed notes on the roof stability were also taken at each location. Note that all observations were made in supported entries.

4.6.1 The CMRR

During the course of this project, the author had the opportunity to visit with Chris Mark who developed the CMRR and co-authored the original CMRR publication (Molinda and Mark, 1994). The author was able to learn many of the subtle details of the CMRR data collection process from Chris Mark, in particular how to interpret the ball peen hammer test to obtain the unit UCS and how to interpret the unit discontinuity cohesion through the splitting test with a mason chisel. These parameters are arguably subjective, and the accuracy of the determination of these parameters by the author was likely improved having learned directly from one of the researchers who developed the CMRR.

The CMRR component values (discontinuity intensity, discontinuity shear strength etc.) were recorded using the methods outlined by Molinda and Mark (1994).

UCS was estimated underground by means of the indentation test by striking the exposed rock face with the round end of a ball peen hammer. The impact reaction was
then compared to the chart shown in Figure 4.3. The strength value from Figure 13 was then assigned a CMRR rating value using Table 4.1.

![Diagram of ball peen hammer impact test results and their correlation to uniaxial compressive strength.](image)

Figure 4.3: Ball peen hammer impact test results and their correlation to uniaxial compressive strength. (Molinda and Mark, 1994).

<table>
<thead>
<tr>
<th>Strength (MPa (psi))</th>
<th>CMRR Rating</th>
</tr>
</thead>
<tbody>
<tr>
<td>&gt;103 (&gt;15,000)</td>
<td>30</td>
</tr>
<tr>
<td>55-103 (8,000-15,000)</td>
<td>22</td>
</tr>
<tr>
<td>21-55 (3,000-8,000)</td>
<td>15</td>
</tr>
<tr>
<td>7-21 (1,000-3,000)</td>
<td>10</td>
</tr>
<tr>
<td>&lt;7 (&lt;1000)</td>
<td>5</td>
</tr>
</tbody>
</table>

Table 4.1: CMRR Strength – Rating conversion (after Molinda and Mark, 1994).

Discontinuity cohesion was evaluated underground through a splitting test with a mason chisel. The number of chisel blows required to split the rock along the discontinuity planes is correlated to the cohesion as shown in Table 4.2. Discontinuity roughness was estimated visually underground and classified as jagged, wavy or planar...
through comparison with the published rating chart shown in Figure 4.4. The cohesion-roughness rating was then calculated using Table 4.3.

**Table 4.2:** Discontinuity cohesion classification from no. of hammer blows required to split bedding (Molinda and Mark, 1994).

<table>
<thead>
<tr>
<th>Number of Hammer Blows required to split bedding with a mason chisel</th>
<th>Cohesion classification</th>
</tr>
</thead>
<tbody>
<tr>
<td>&gt;7</td>
<td>Strong</td>
</tr>
<tr>
<td>4-7</td>
<td>Moderate</td>
</tr>
<tr>
<td>1-3</td>
<td>Weak</td>
</tr>
<tr>
<td>0</td>
<td>Slickensided</td>
</tr>
</tbody>
</table>

**Figure 4.4:** Guide for estimating discontinuity roughness (Molinda and Mark, 1994).

**Table 4.3:** CMRR discontinuity cohesion and roughness rating value (Molinda and Mark, 1994).

<table>
<thead>
<tr>
<th>Roughness</th>
<th>Strong Cohesion</th>
<th>Moderate Cohesion</th>
<th>Weak Cohesion</th>
<th>Slickensided</th>
</tr>
</thead>
<tbody>
<tr>
<td>Jagged</td>
<td>35</td>
<td>29</td>
<td>24</td>
<td>10</td>
</tr>
<tr>
<td>Wavy</td>
<td>35</td>
<td>27</td>
<td>20</td>
<td>10</td>
</tr>
<tr>
<td>Planar</td>
<td>35</td>
<td>25</td>
<td>16</td>
<td>10</td>
</tr>
</tbody>
</table>
Discontinuity spacing and persistence were evaluated through observation and the discontinuity spacing and persistence rating calculated using Table 4.4.

Table 4.4: CMRR discontinuity spacing and persistence rating value (Molinda and Mark, 1994).

<table>
<thead>
<tr>
<th>Persistence (m(ft))</th>
<th>Spacing (m (ft))</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>&gt;1.8 (&gt;6ft)</td>
</tr>
<tr>
<td>0-0.9 (0-3 ft)</td>
<td>35</td>
</tr>
<tr>
<td>0.9-3 (3-10 ft)</td>
<td>3</td>
</tr>
<tr>
<td>3-9 (10-30 ft)</td>
<td>30</td>
</tr>
<tr>
<td>&gt;9 (&gt;30 ft)</td>
<td>30</td>
</tr>
</tbody>
</table>

Moisture sensitivity was estimated through an immersion test using the data collection sheet shown in Figure 4.5 and visually underground. Table 4.5 was then used to assign a rating value to the moisture sensitivity.

Table 4.5: CMRR moisture sensitivity rating value (Molinda and Mark, 1994).

<table>
<thead>
<tr>
<th>Moisture Sensitivity</th>
<th>Rating</th>
</tr>
</thead>
<tbody>
<tr>
<td>Not sensitive</td>
<td>0</td>
</tr>
<tr>
<td>Slightly sensitive</td>
<td>-3</td>
</tr>
<tr>
<td>Moderately sensitive</td>
<td>-10</td>
</tr>
<tr>
<td>Severely sensitive</td>
<td>-25</td>
</tr>
</tbody>
</table>

The presence of a strong bed on the bolted interval was estimated from logs, where available, and the corresponding adjustment factor applied. If no logs were available for the strata above the measurement site, this parameter was inferred from mine maps and discussion with the mine personnel. Similarly, the number of units in the immediate roof were estimated from logs (if available), mine maps and talking with mine
personnel. The groundwater adjustment was based on observations and the surcharge effect was estimated using the same methods as for the number of units in the roof. The CMRR was then calculated for each data collection site. For each unit in the roof, the unit rating was first calculated by summing the discontinuity cohesion - roughness rating and the discontinuity spacing - persistence rating. If there were multiple discontinuities e.g. bedding, slickensides etc. in the unit, this was done for each one individually. If there was more than one discontinuity set, a multiple discontinuity adjustment was also applied to the lowest discontinuity rating, as shown in Table 4.6. This discontinuity rating was then added to the unit CMRR value. The moisture sensitivity value was then applied only to the lowermost unit of the roof. This procedure was repeated for all units in the bolted interval.

Table 4.6: CMRR multiple discontinuity adjustment values (Molinda and Mark, 1994).

<table>
<thead>
<tr>
<th>Two lowest individual discontinuity ratings both lower than -</th>
<th>Adjustment</th>
</tr>
</thead>
<tbody>
<tr>
<td>30</td>
<td>-5</td>
</tr>
<tr>
<td>40</td>
<td>-4</td>
</tr>
<tr>
<td>50</td>
<td>-2</td>
</tr>
</tbody>
</table>

Where there was more than one unit in the bolted interval, each of the previously calculated unit ratings was multiplied by the estimated thickness of the relevant unit. The output values were then summed and the total divided by the length of the bolted interval. The strong bed adjustment, the adjustment for number of units, groundwater
**Figure 4.5: Data Collection sheet for the CMRR immersion test (Molinda and Mark, 2005).**

**IMMERSION TEST**

<table>
<thead>
<tr>
<th>Mine: __________________________</th>
<th>Date: __________________________</th>
</tr>
</thead>
<tbody>
<tr>
<td>Unit No.: __________________________</td>
<td>Tester: __________________________</td>
</tr>
<tr>
<td>Sample Description (Lithology, bedding, etc.):</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th><strong>Immersion</strong></th>
<th><strong>Breakability</strong></th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Observation</strong></td>
<td><strong>Rating</strong></td>
</tr>
<tr>
<td>Appearance of Water</td>
<td></td>
</tr>
<tr>
<td>Clear = 0</td>
<td></td>
</tr>
<tr>
<td>Misty = -1</td>
<td></td>
</tr>
<tr>
<td>Cloudy = -3</td>
<td></td>
</tr>
<tr>
<td>Talus Formation</td>
<td>Breakability Index</td>
</tr>
<tr>
<td>None = 0</td>
<td></td>
</tr>
<tr>
<td>Minor = -1</td>
<td></td>
</tr>
<tr>
<td>Major = -3</td>
<td></td>
</tr>
<tr>
<td>Cracking of Sample</td>
<td></td>
</tr>
<tr>
<td>None = 0</td>
<td></td>
</tr>
<tr>
<td>Minor/Random = -1</td>
<td></td>
</tr>
<tr>
<td>Major/Preferred Orientation = -3</td>
<td></td>
</tr>
<tr>
<td>Specimen Breakdown = -9</td>
<td></td>
</tr>
</tbody>
</table>

**Total Immersion Index** |

**Procedure for Immersion Test**
1. Select sample(s) - ~ hand sized.
2. Test for hand breakability.
3. Rinse specimen (to remove surface dirt, dust, etc.).
4. Immerse in water for 1 hour.
5. Observe and rate water appearance, talus formation, and cracking of sample.
7. Retest for hand breakability.
8. *Determine Breakability Index.*
9. The final Immersion Test Index is the greater of the Breakability Index or the Immersion Index.
adjustment and surcharge adjustments were then applied to this number to get the final CMRR at a given location.

4.6.2 Roof Stability

Detailed notes on the roof stability were made at each site. Initially this parameter was interpreted as binary (stable or unstable), but this was later deemed inappropriate, as roof stability exists as a continuum of possible states in reality. The end result was to classify roof stability on a scale of 1-4 with 4 being the most stable and 1 the least. A more detailed discussion on roof stability and its classification may be found in Chapter 6.2.

Four categories were chosen based on the level of detail in the data recorded. It should be noted that many criteria beyond those used could be considered in the classification of roof stability, and it may be appropriate to add to or amend the criteria used in this study for the purposes of future research. The following criteria describe each of the four categories:

1. Previous Failure: domed area left behind.
2. Significant Skin Failure: approx. 20 or more hand sized pieces in 1 square meter of mesh and/or visible cracking present.
3. Minor Skin Failure: less than 20 hand sized pieces in 1 square meter of mesh.
4. No evidence of Skin Failure

It should be noted that while the above scale was considered the most appropriate approach, the analyses described in Chapter 4.7 were performed
considering roof stability both as a scale and as a binary parameter. It should also be noted that all of the data was collected in areas with roof support installed. This is a confounding influence as a more supported roof is likely more stable than if the support were not there, and would likely have less evidence of failure. Conversely, a highly supported area, in anticipation of poor roof conditions, could make the data collector assume the area to be more unstable than it may be in reality, particularly if the mine engineers were conservative. This analysis ignored the previously installed support and roof support classifications were based solely on the criteria above. Using the above criteria is considered to allow roof stability to be evaluated as effectively as possible in the circumstances. Nonetheless, the influence of previous support should be acknowledged as a potential source of error, particularly as it varies throughout the mines, and the results treated accordingly.

4.6.3 Additional Parameters

The depth of cover and slope grade of surface topography were obtained from pre-existing mine maps. The depth of cover was estimated based on mine maps specifically showing depth of cover contours. Where the data collection location was between contours, inference was used, with the assumption that the change in depth of cover was linear between contours.

The slope was calculated using one contour width as the horizontal distance and the vertical distance was calculated using the contour lines. For consistency, the horizontal distance was always measured to the next downhill contour and perpendicular to the contour lines. This does however, assume the slope to be the
same between the contours, which may or may not be the case in reality depending on the resolution of the map used. Another method considered was to calculate several slopes at a site and find the minimum horizontal distance for each of the calculated slopes to be within one standard deviation of each other. Doing this at a small number (3-5) sites would then give a minimum horizontal distance for calculating slope which could be extrapolated across the whole map. However, this method also requires inference between the contour lines so it was not pursued.

Because the change in slope may be more influential on ground control and mine roof stability than the instantaneous slope value, the second derivative of topography was also calculated for each data collection location at Mine B. This allowed the change in slope above each of the data collection locations to be evaluated. The topographic relief in the maps available at Mine A was not sufficiently detailed to carry out such an analysis. At Mine B, this was done by selecting a line segment at each of the data collection locations. This is illustrated in Figure 4.6. The length of the line was taken to be the typical thickness of a topographic feature in the area – this allowed for adaptability when the contour lines had different spacing. The orientation of these lines was always taken perpendicular to the contour lines to determine the surface curvature in the direction of maximum instantaneous slope. Along the line, coordinates were taken at four points – either where the lines crossed contours or at the ends of the lines. This then generated multiple line segments within each of the lines for which a slope could be calculated (see Figure 4.6).

The three slope estimates were then plotted against the center of their associated range in the x direction. The gradient of this best fit line is the topographic
curvature or the approximate second derivative of the topography (first derivation of the slope). For each site, curvature values were therefore generated and were then evaluated for correlation with roof stability. At some sites it would have been possible to draw several lines to evaluate topography at different scales, but at other locations doing so would have led to significant inference in the lines due to the widely spaced contours. The line length (the typical thickness of a topographic feature in the area) used was found to be on the same order of magnitude as what might be considered an area of surface influence of mining based on an angle of draw of 20°.

Figure 4.6: An illustration of the method used to calculate the second derivative of topography (curvature) at Mine B.

Data pertaining to the other non-CMRR parameters, if applicable, were recorded individually at each location. All of the other non-CMRR parameters were recorded as
binary. For example, the data collection site was either located in an intersection or it was not. Certain non-CMRR parameters, such as the rib and floor condition, which was recorded as stable or unstable, could benefit from being on a scale of stability like the roof stability parameter. Unfortunately, there was insufficient detail in the data recorded to sufficiently delineate this parameter into categories. It is however, something to consider in future work.

For the coal seam merge at Mine B, the parameter was recorded as ‘yes’ where the rider coal seam was between 4 and 12 ft above the main coal seam. This is where it was considered to be problematic. Where the rider coal was in the ribs or more than 12 ft above the main coal seam, the parameter was recorded as ‘no’.

As much information as possible was recorded at each measurement location for the geology, stratigraphy, roof stability and site-specific ground control issues. Detailed notes were taken in case revisions were required later in the data analysis process. This meant the data could be referred to at a later date without the need to return to the sites.

This fieldwork was carried out in Fall 2017 and Spring 2018 and formed the second and final visits to Mines A and B for the purposes of this project.

4.7 Data Analysis

A summary of the analyses performed and on which variables (at Mines A and B) are provided in Tables 4.7 and 4.8 respectively. A discussion of the information presented is provided below.
Table 4.7: Summary of the analyses performed on the data from Mine A.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Evaluated with</th>
<th>Statistical Tests Used</th>
</tr>
</thead>
<tbody>
<tr>
<td>CMRR</td>
<td>Roof Stability (Binary)</td>
<td>ANOVA Logistic Regression</td>
</tr>
<tr>
<td></td>
<td>Roof Stability (1-4 Scale)</td>
<td>ANOVA Logistic Regression</td>
</tr>
<tr>
<td></td>
<td>Faulting</td>
<td>ANOVA</td>
</tr>
<tr>
<td></td>
<td>Intersection Location</td>
<td>ANOVA</td>
</tr>
<tr>
<td></td>
<td>Sandstone Channel</td>
<td>ANOVA</td>
</tr>
<tr>
<td></td>
<td>Depth of Cover</td>
<td>Linear Regression</td>
</tr>
<tr>
<td></td>
<td>Slope Angle of Surface Topography</td>
<td>Linear Regression</td>
</tr>
<tr>
<td></td>
<td>Faulting</td>
<td>Fisher Exact Test Spearman</td>
</tr>
<tr>
<td></td>
<td>Intersection Location</td>
<td>Fisher Exact Test Spearman</td>
</tr>
<tr>
<td></td>
<td>Sandstone Channel</td>
<td>Fisher Exact Test Spearman</td>
</tr>
<tr>
<td></td>
<td>Depth of Cover</td>
<td>ANOVA Spearman</td>
</tr>
<tr>
<td></td>
<td>Slope Angle of Surface Topography</td>
<td>ANOVA Spearman</td>
</tr>
<tr>
<td></td>
<td>Faulting</td>
<td>Chi-squared test Spearman</td>
</tr>
<tr>
<td></td>
<td>Intersection Location</td>
<td>Chi-squared test Spearman</td>
</tr>
<tr>
<td></td>
<td>Sandstone Channel</td>
<td>Chi-squared test Spearman</td>
</tr>
<tr>
<td></td>
<td>Depth of Cover</td>
<td>ANOVA Spearman</td>
</tr>
<tr>
<td></td>
<td>Slope Angle</td>
<td>ANOVA Spearman</td>
</tr>
<tr>
<td></td>
<td>Roof Stability (Binary)</td>
<td>ANOVA</td>
</tr>
</tbody>
</table>

CMRR (no cohesion and roughness parameter)
CMRR (no spacing or persistence parameter)
CMRR (no strength parameter)
Table 4.7: (continued)

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Evaluated with</th>
<th>Statistical Tests Used</th>
</tr>
</thead>
<tbody>
<tr>
<td>CMRR (no moisture sensitivity parameter)</td>
<td>Roof Stability (1-4 Scale)</td>
<td>ANOVA Spearman</td>
</tr>
<tr>
<td>CMRR (no cohesion and roughness parameter)</td>
<td></td>
<td></td>
</tr>
<tr>
<td>CMRR (no spacing or persistence parameter)</td>
<td></td>
<td></td>
</tr>
<tr>
<td>CMRR (no strength parameter)</td>
<td></td>
<td></td>
</tr>
<tr>
<td>CMRR (no moisture sensitivity parameter)</td>
<td></td>
<td></td>
</tr>
<tr>
<td>CMRR, Depth of Cover</td>
<td>Roof Stability (1-4 Scale)</td>
<td>Logistic Regression</td>
</tr>
<tr>
<td>CMRR, Slope Angle</td>
<td>Roof Stability (1-4 Scale)</td>
<td>Logistic Regression</td>
</tr>
<tr>
<td>CMRR, Depth of Cover, Slope Angle</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Depth of Cover</td>
<td>Roof Stability (1-4 Scale)</td>
<td>Logistic Regression</td>
</tr>
<tr>
<td>Slope Angle</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Table 4.8: Summary of the analyses performed on the data from Mine B.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Evaluated with</th>
<th>Statistical Tests Used</th>
</tr>
</thead>
<tbody>
<tr>
<td>CMRR</td>
<td>Roof Stability (Binary)</td>
<td>ANOVA Logistic Regression</td>
</tr>
<tr>
<td></td>
<td>Roof Stability (1-4 Scale)</td>
<td>ANOVA Logistic Regression</td>
</tr>
<tr>
<td></td>
<td>Faulting</td>
<td>ANOVA</td>
</tr>
<tr>
<td></td>
<td>Intersection Location</td>
<td>ANOVA</td>
</tr>
<tr>
<td></td>
<td>Coal Seam Merge</td>
<td>ANOVA</td>
</tr>
<tr>
<td></td>
<td>Depth of Cover</td>
<td>Linear Regression</td>
</tr>
<tr>
<td></td>
<td>Slope Angle of Surface Topography</td>
<td>Linear Regression</td>
</tr>
<tr>
<td></td>
<td>Curvature of Surface Topography</td>
<td>Linear Regression</td>
</tr>
</tbody>
</table>

90
Table 4.8: (continued)

<table>
<thead>
<tr>
<th>Roof Stability (Binary)</th>
<th>Rib/Floor Condition</th>
<th>Fisher Exact Test Spearman</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Intersection Location</td>
<td>Fisher Exact Test Spearman</td>
</tr>
<tr>
<td></td>
<td>Coal Seam Merge</td>
<td>Fisher Exact Test Spearman</td>
</tr>
<tr>
<td></td>
<td>Depth of Cover</td>
<td>ANOVA Spearman</td>
</tr>
<tr>
<td></td>
<td>Slope Angle of Surface Topography</td>
<td>ANOVA Spearman</td>
</tr>
<tr>
<td></td>
<td>Curvature of Surface Topography</td>
<td>ANOVA Spearman</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Roof Stability (1-4 Scale)</th>
<th>Rib and Floor Condition</th>
<th>Chi-squared Test Spearman</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Intersection Location</td>
<td>Chi-Squared Test Spearman</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Roof Stability (1-4 Scale)</th>
<th>Coal Seam Merge</th>
<th>Chi-Squared Test Spearman</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Depth of Cover</td>
<td>ANOVA Spearman</td>
</tr>
<tr>
<td></td>
<td>Slope Angle of Surface Topography</td>
<td>ANOVA Spearman</td>
</tr>
</tbody>
</table>

| Roof Stability (1-4 Scale) | Curvature of Surface Topography | ANOVA |

| CMRR (no cohesion and roughness parameter) | Roof Stability (Binary) | ANOVA |
| CMRR (no spacing or persistence parameter) | Roof Stability (1-4 Scale) | ANOVA Spearman |
| CMRR (no strength parameter) | Roof Stability (1-4 Scale) | ANOVA Spearman |
| CMRR (no moisture sensitivity parameter) | Roof Stability (1-4 Scale) | ANOVA Spearman |
| CMRR (no cohesion and roughness parameter) | Roof Stability (1-4 Scale) | ANOVA Spearman |
| CMRR (no spacing or persistence parameter) | Roof Stability (1-4 Scale) | ANOVA Spearman |
| CMRR (no strength parameter) | Roof Stability (1-4 Scale) | ANOVA Spearman |
| CMRR (no moisture sensitivity parameter) | Roof Stability (1-4 Scale) | ANOVA Spearman |
The data were first analyzed for correlation between the CMRR and roof stability. A strong correlation at both mines would mean that the CMRR likely includes the most influential parameters which are predictive of roof instability at Mines A and B. Little correlation between CMRR and roof stability suggests there are other parameters which influence roof instability at Mines A and B and which merit further investigation. To give an initial sense of the correlation between the CMRR and recorded roof stability, the
CMRR values for stable roof were plotted and compared to a plot of CMRR values for unstable roof. This was done for the data at both mines. If all CMRR values are high for stable roof and low for unstable roof, this would suggest the CMRR is comprehensive and effective (at Mines A and B). If there is significant scatter in the values, it would suggest they are not strongly correlated and that other parameters influence roof instability. It is worth noting here that only two mines are being used as case studies; this study aims to provide a starting point in evaluating if the CMRR is or is not effective at coal mines in the Western US and an indication of how it may be improved. Making any definitive big picture conclusions, however, requires a much larger data set.

To quantitatively evaluate the correlation between the CMRR and roof stability, the Analysis of Variance (ANOVA) method was used to test the null hypothesis that the CMRR does not vary as a function of roof stability category and calculate a probability (p) value. For a p value < 0.05 the null hypothesis may be rejected with 95% confidence. As part of the ANOVA process, a comparative boxplot of the data showing the spread of CMRR values for each roof stability category is produced which allows for a visual inspection of the data. This ANOVA analysis was carried out for the roof stability as a binary parameter and on a scale of 1-4. ANOVA was also used to evaluate correlation and obtain p values between the CMRR and the binary non-CMRR parameters collected. The continuous non-CMRR parameters such as depth of cover were evaluated for correlation with the CMRR using linear regression (e.g. CMRR vs depth of cover, CMRR vs location at an intersection, etc.).

The correlation between roof stability and each parameter recorded was evaluated next. For the analysis using the binary interpretation of roof stability, ANOVA
was used to analyze the correlation with depth of cover and surface topography as 
these parameters were recorded as continuous numerical values. ANOVA cannot 
evaluate the correlation between two categorical variables, however. In order to 
examine the correlation between roof stability when recorded as stable or unstable and 
other non-CMRR parameters which were also recorded as binary (e.g. sandstone 
channel presence as yes or no), the Fisher Exact Test was used (Freeman and 
Campbell, 2007). The fisher test evaluates the null hypothesis that there is no 
association between the independent and dependent variables. If the $p$ value generated 
is < 0.05, the null hypothesis may be rejected at 95% confidence.

Where the roof stability scale was analyzed with the recorded non-CMRR 
parameters, ANOVA was again used to analyze the correlation with depth of cover and 
slope of surface topography. The ANOVA analysis assumes one of the variables in the 
analysis to be continuous (the 1-4 scale is not continuous) however, so this method was 
not applicable to examine the correlation between the roof stability scale and the binary 
non-CMRR parameters such as faulting. The Fisher Exact Test uses a 2 x 2 matrix i.e. 
the two variables can only have two categories each so it was not appropriate to use 
either. The Chi squared test was therefore used to evaluate the correlation between roof 
stability scale and the non-CMRR parameters recorded as binary. This test can be used 
to evaluate correlation between two categorical variables as it tests whether there is a 
significant difference between the expected frequencies and the observed frequencies 
in one or more categories. The expected frequencies are equal distribution between the 
categories. The statistical significance ($p$ value) of the Chi squared test was also 
calculated.
Next, the CMRR values at each location were recalculated for all but one of the four main component parameters: discontinuity cohesion and roughness, discontinuity spacing and persistence, strength or moisture sensitivity. This was done by normalizing the adjusted values to the maximum available number of points. Due to the structure of the CMRR, the discontinuity cohesion parameter cannot be removed by itself as the corresponding rating depends on the roughness too. The same applies to the discontinuity spacing and persistence parameter. The adjustment factors were not removed or further analyzed as they were only applied in a small number of cases (<5).

The correlation between these ‘adjusted’ CMRRs and roof stability and the correlation between the ‘adjusted’ CMRRs and the non-CMRR parameters recorded were then evaluated. This gives an indication of how influential each constituent in the CMRR is with respect to roof stability (at Mines A and B).

The above analyses were also run using the Spearman correlation test and logistic regression where applicable. When the input data did not match that required for a given test, the test was not used. An example of this is the logistic regression analysis which predicts the probability that a continuous predictor variable will fall into a given category. This therefore works well in evaluating the predictive capacity of the CMRR on roof stability but does not work for predicting the CMRR from a binary variable such as faulting.

The Spearman test determines the strength and direction of the monotonic relationship between two variables, essentially drawing a line of best fit through the data and measuring how far the data points are from this line. The two variables need to be measured on an ordinal, interval, or a ratio scale, but do not need to be measured on
the same scale. The Spearman probability \((p)\) value is a measure of how likely it is a point will fall on the best fit line. As ordinal data is an acceptable input the Spearman test may be used where a roof stability scale of 1-4 is one of the variables.

Logistic regression calculates a model for the relative ‘risk’ of a parameter being in one category versus another and ultimately the probability that a given value of a continuous variable would fall into a certain category. The logistic regression model intercept and coefficients are calculated first and their statistical significance can then be calculated using various methods. The analyses for this research were all carried out in Matlab and the software MiniTab which use the T-test to calculate the probability \((p)\) values of the model coefficients being statistically significant. More importantly however, logistic regression also returns a probability value which is of practical significance (for example, if the CMRR is a known value, the probability that the roof is unstable can be estimated). This analysis is therefore particularly valuable. The ANOVA and Spearman analyses only give the statistical significance of the correlation, however statistical significance does not imply engineering or practical significance. Statistical significance simply allows conclusions to be drawn about the strength of a correlation. For example: if there were 1000 measurements of CMRR in a data set and the CMRR values in the unstable cases ranged from 15-45 with an average of 30 and the CMRR values in the stable cases ranged from 20-50 with an average of 35, the ANOVA analysis would return a small \(p\) value due to the size of the data set. However, using logistic regression to predict the roof stability for a CMRR of 30 may give a 45% chance of it being stable and a 55% that it is unstable. In such a case, the statistically significant relationship is of limited practical use. Logistic regression was therefore used to evaluate the probability
of the CMRR falling in a given roof stability category to provide a practically useable result. The analysis was carried out for both binary roof stability and the roof stability scale. To further evaluate the predictive effect of the other continuous non-CMRR parameters such as depth of cover, slope angle and topographic curvature on roof stability, logistic regression analyses were carried out using combinations of these variables along with the CMRR as input predictor variables. Logistic regression analyses were also run using the continuous non-CMRR parameters individually as inputs. These additional logistic regression analyses were only carried out with the roof condition scale. As discussed previously and in Chapter 6.2, the roof condition scale is considered to be more representative of reality and thereby provide more useful results.

Additional analyses were also run. Location at a valley base was analyzed for correlation with roof stability using the Fisher Exact Test and the Chi Squared Test. An examination of the data collection sites with respect to loading condition was also carried out. To evaluate whether the support used at the mines was a proxy for roof stability and highlight how much of a confounding influence pre-installed support could be when evaluating roof stability, the non-CMRR parameters collected at Mine B were analyzed with support category. This was only done at Mine B because there were a series of systematic categories of support used and a map showing which support was used where was available. At Mine A, a map of primary support installed was not available and discussion with the mine engineers indicated the high variability in support throughout the mine. At Mine B, the four ground control categories used to determine support requirements are shown in Table 4.9.
Table 4.9: Ground control criteria used to determine support requirements at Mine B.

<table>
<thead>
<tr>
<th>Support Category Number</th>
<th>Depth of Cover</th>
<th>Slope</th>
<th>Rider Seam (distance from the immediate roof)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Shallow (&lt;600 ft)</td>
<td>Shallow (&lt;60%)</td>
<td>&lt; 2 ft or &gt;12 ft</td>
</tr>
<tr>
<td>2</td>
<td>Deep (&lt;600 ft)</td>
<td>Steep (&gt;60%)</td>
<td>&lt; 2 ft or &gt;12 ft</td>
</tr>
<tr>
<td>3</td>
<td>Deep (&lt;600 ft)</td>
<td>Steep (&gt;60%)</td>
<td>&gt; 2 ft or &lt;12 ft</td>
</tr>
<tr>
<td>4</td>
<td>Shallow (&lt;600 ft)</td>
<td>Shallow (&lt;60%)</td>
<td>&gt; 2 ft or &lt;12 ft</td>
</tr>
</tbody>
</table>

The support for Category 1 was 6 ft fully grouted bolts on a 5 ft spacing with wire mesh were used in the entries. At intersections, 7 ft fully grouted bolts on 5 ft spacing were used, also with wire mesh. The exact support used in categories 2, 3 and 4 varies depending on whether the location in question is in the East Mains, dual tailgate return or single tailgate return. Information on the exact support used for each of these categories is attached in Appendix A.

It is expected that there will be non-CMRR parameters which show a strong correlation with roof stability, but not with the CMRR. If a certain parameter is strongly correlated with roof stability at both mines and not included in the CMRR, it may be recommended that future work be carried out to include data from other mines and ultimately to see if it is possible to include the parameter in the CMRR.
5.1 Introduction

The results of the methods set out in Chapter 4 are presented here for both Mine A and Mine B.

5.2 Correlation between the CMRR and Roof Stability

The initial classification of roof stability at Mine A and Mine B was binary (stable or unstable). This was later changed to be on a scale of 1-4. The reasons for this are discussed in Chapter 6.2.

5.2.1 Mine A

At Mine A, the CMRR values calculated ranged from 16 to 47. Figure 5.1 shows the results from the ANOVA analysis evaluating correlation between the CMRR and roof stability as a binary parameter at Mine A. The $p$ value for the analysis was calculated to be 0.20. The $p$ value for ANOVA analysis evaluating the correlation between CMRR and roof stability on a scale of 1-4, was 0.0073. The boxplot results from the ANOVA analysis evaluating the correlation between CMRR and roof stability is shown in Figure 5.2.
Figure 5.1: Comparative boxplot result from the ANOVA analysis of CMRR and binary roof stability; $p = 0.20$. The $n$ values indicate the number of data points in each category.

Figure 5.2: Comparative boxplot result from the ANOVA analysis of CMRR and roof stability on a scale of 1-4; $p = 0.0073$. The $n$ values indicate the number of data points in each category.
5.2.2 Mine B

At Mine B, the CMRR values calculated ranged from 29 to 62. Figure 5.3 shows the results from the ANOVA analysis evaluating correlation between the CMRR and roof stability as a binary parameter at Mine B. The $p$ value for the analysis was calculated to be 0.24. The $p$ value for ANOVA analysis evaluating the correlation between CMRR and roof stability on a scale of 1-5 at Mine B was found to be 0.53. When this was changed to a scale of 1-4, the $p$ value increased to 0.95. The boxplot result from the ANOVA analysis evaluating the correlation between CMRR and roof stability is shown in Figure 5.4.

Figure 5.3: Comparative boxplot result from the ANOVA analysis of CMRR and binary roof stability; $p = 0.24$. The $n$ values indicate the number of data points in each category.
5.3 Correlation Between the CMRR and non-CMRR Parameters Collected

5.3.1 Mine A

The results of the ANOVA analyses evaluating correlation between the CMRR at each location and each of the non-CMRR parameters collected at Mine A are presented in Table 5.1.

Table 5.1: Summary of \( p \) values for the ANOVA analysis between the CMRR and the non-CMRR parameters collected at each site at Mine A.

<table>
<thead>
<tr>
<th>Parameter evaluated with CMRR</th>
<th>( p ) Value from ANOVA</th>
<th>( p ) Value from Linear Regression</th>
<th>Number of Data Points Evaluated</th>
</tr>
</thead>
<tbody>
<tr>
<td>Faulting</td>
<td>0.007</td>
<td>--</td>
<td>Yes = 7, No = 23</td>
</tr>
<tr>
<td>Sandstone Channel</td>
<td>0.23</td>
<td>--</td>
<td>Yes = 13, No = 17</td>
</tr>
<tr>
<td>Intersections</td>
<td>0.005</td>
<td>--</td>
<td>Yes = 13, No = 17</td>
</tr>
<tr>
<td>Depth of Cover</td>
<td>--</td>
<td>0.81</td>
<td>30</td>
</tr>
<tr>
<td>Surface Topography</td>
<td>--</td>
<td>0.71</td>
<td>30</td>
</tr>
</tbody>
</table>
5.3.2 Mine B

The results of the ANOVA analyses evaluating correlation between the CMRR at each location and each of the non-CMRR parameters collected at Mine B are presented in Table 5.2.

Table 5.2: Summary of $p$ values for the ANOVA analysis between the CMRR and the non-CMRR parameters collected at each site at Mine B.

<table>
<thead>
<tr>
<th>Parameter Evaluated with CMRR</th>
<th>$p$ Value from ANOVA</th>
<th>$p$ Value from Linear Regression</th>
<th>Number of Data Points Evaluated</th>
</tr>
</thead>
<tbody>
<tr>
<td>Coal Seam Merge</td>
<td>0.0077</td>
<td>--</td>
<td>Yes = 6, No = 24</td>
</tr>
<tr>
<td>Rib/Floor Condition</td>
<td>0.62</td>
<td>--</td>
<td>Unstable = 7, Stable = 23</td>
</tr>
<tr>
<td>Intersection</td>
<td>0.31</td>
<td>--</td>
<td>Yes = 17, No = 13</td>
</tr>
<tr>
<td>Depth of Cover</td>
<td>--</td>
<td>0.31</td>
<td>30</td>
</tr>
<tr>
<td>Curvature of Surface Topography</td>
<td>--</td>
<td>0.05</td>
<td>30</td>
</tr>
<tr>
<td>Slope of Surface Topography</td>
<td>--</td>
<td>0.81</td>
<td>30</td>
</tr>
</tbody>
</table>

5.4 Correlation between Roof Stability and non-CMRR Parameters Collected

5.4.1 Mine A

The results of the analyses evaluating the correlation between roof stability as a binary parameter and the non-CMRR parameters collected at Mine A are presented in Table 5.3. After the roof stability parameter was changed to be on a scale of 1-4, the same analyses with each parameter collected were run again. The results of this are shown in Table 5.4.
Table 5.3: Summary of \( p \) values for the analysis of correlation between roof stability (binary) and the non-CMRR parameters collected at Mine A using the ANOVA analysis and the Fisher Exact Test.

<table>
<thead>
<tr>
<th>Parameter evaluated with Roof Stability (Binary)</th>
<th>( p ) Value from ANOVA</th>
<th>( p ) Value from Fisher Exact Test</th>
<th>( p ) Value from Spearman Test</th>
<th>Number of Data Points Evaluated</th>
</tr>
</thead>
<tbody>
<tr>
<td>Faulting</td>
<td>--</td>
<td>0.34</td>
<td>0.01</td>
<td>Yes = 7, No = 23</td>
</tr>
<tr>
<td>Sandstone Channel</td>
<td>--</td>
<td>0.69</td>
<td>0.44</td>
<td>Yes = 13, No = 17</td>
</tr>
<tr>
<td>Intersections</td>
<td>--</td>
<td>0.69</td>
<td>0.77</td>
<td>Yes = 13, No = 17</td>
</tr>
<tr>
<td>Depth of Cover</td>
<td>0.44</td>
<td>--</td>
<td>0.95</td>
<td>30</td>
</tr>
<tr>
<td>Slope of Surface Topography</td>
<td>0.71</td>
<td>--</td>
<td>0.49</td>
<td>30</td>
</tr>
</tbody>
</table>

Table 5.4: Summary of \( p \) values for the analysis of correlation between roof stability (1-4 scale) and the non-CMRR parameters collected at Mine A from the ANOVA analysis and the Chi Squared Test.

<table>
<thead>
<tr>
<th>Parameter evaluated with Roof Stability (1-4 scale)</th>
<th>( p ) Value from ANOVA</th>
<th>( p ) Value from Chi Squared test</th>
<th>( p ) value from Spearman Test</th>
<th>Number of Data Points Evaluated</th>
</tr>
</thead>
<tbody>
<tr>
<td>Faulting</td>
<td>--</td>
<td>0.014</td>
<td>0.0013</td>
<td>Yes = 7, No = 23</td>
</tr>
<tr>
<td>Sandstone Channel</td>
<td>--</td>
<td>0.32</td>
<td>0.55</td>
<td>Yes = 13, No = 17</td>
</tr>
<tr>
<td>Intersections</td>
<td>--</td>
<td>0.52</td>
<td>0.59</td>
<td>Yes = 13, No = 17</td>
</tr>
<tr>
<td>Depth of Cover</td>
<td>0.08</td>
<td>--</td>
<td>0.53</td>
<td>30</td>
</tr>
<tr>
<td>Slope of Surface Topography</td>
<td>0.07</td>
<td>--</td>
<td>0.16</td>
<td>30</td>
</tr>
</tbody>
</table>

5.4.2 Mine B

The results of the analyses evaluating the correlation between roof stability as a binary parameter and the non-CMRR parameters collected at Mine B are presented in
Table 5.5: The same analyses were run again after changing the roof stability parameter to be on a scale of 1-4. The results of this are shown in Table 5.6.

Table 5.5: Summary of \( p \) values for the analysis of correlation between roof stability (binary) and the non-CMRR parameters collected at Mine B from the ANOVA analysis and the Fisher Exact Test.

<table>
<thead>
<tr>
<th>Parameter Evaluated with Roof Stability</th>
<th>( p ) Value from ANOVA</th>
<th>( p ) Value from Fisher Exact Test</th>
<th>( p ) value from Spearman Test</th>
<th>Number of Data Points Evaluated</th>
</tr>
</thead>
<tbody>
<tr>
<td>Coal Seam Merge</td>
<td>--</td>
<td>0.36</td>
<td>0.2104</td>
<td>Yes = 6, No = 24</td>
</tr>
<tr>
<td>Rib/Floor Condition</td>
<td>--</td>
<td>0.19</td>
<td>0.0924</td>
<td>Unstable = 7, Stable = 23</td>
</tr>
<tr>
<td>Intersections</td>
<td>--</td>
<td>0.071</td>
<td>0.0525</td>
<td>Yes = 17, No = 13</td>
</tr>
<tr>
<td>Depth of Cover</td>
<td>0.26</td>
<td>--</td>
<td>0.328</td>
<td>30</td>
</tr>
<tr>
<td>Curvature of Surface Topography</td>
<td>0.069</td>
<td>--</td>
<td>0.352</td>
<td>30</td>
</tr>
<tr>
<td>Slope of Surface Topography</td>
<td>0.016</td>
<td>--</td>
<td>0.0333</td>
<td>30</td>
</tr>
</tbody>
</table>

Table 5.6: Summary of \( p \) values for the analysis of correlation between roof condition (1-4 scale) and the non-CMRR parameters collected at Mine B from the ANOVA analysis and the Chi Squared Test.

<table>
<thead>
<tr>
<th>Parameter Evaluated with Roof Stability</th>
<th>( p ) Value from ANOVA</th>
<th>( p ) Value from Chi Squared Test</th>
<th>( p ) Value from Spearman test</th>
<th>Number of Data Points Evaluated</th>
</tr>
</thead>
<tbody>
<tr>
<td>Coal Seam Merge</td>
<td>--</td>
<td>0.25</td>
<td>0.05</td>
<td>Yes = 6, No = 24</td>
</tr>
<tr>
<td>Rib/Floor Condition</td>
<td>--</td>
<td>0.079</td>
<td>0.17</td>
<td>Unstable = 7, Stable = 23</td>
</tr>
<tr>
<td>Intersections</td>
<td>--</td>
<td>0.054</td>
<td>0.0079</td>
<td>Yes = 17, No = 13</td>
</tr>
<tr>
<td>Depth of Cover</td>
<td>0.024</td>
<td>--</td>
<td>0.53</td>
<td>30</td>
</tr>
<tr>
<td>Curvature of Surface Topography</td>
<td>0.34</td>
<td>--</td>
<td>0.55</td>
<td>30</td>
</tr>
<tr>
<td>Slope of Surface Topography</td>
<td>0.69</td>
<td>--</td>
<td>0.11</td>
<td>30</td>
</tr>
</tbody>
</table>
5.5 Correlation Between Adjusted CMRR Values and Roof Stability

5.5.1 Mine A

The four primary components of the CMRR are joint cohesion and roughness, joint spacing and persistence, strength and moisture sensitivity. These components were removed in turn from the original CMRR values and four adjusted CMRR values were calculated for each data collection location. The adjusted CMRR values were analyzed for correlation with roof stability (both as a binary parameter and on a scale of 1-4) at Mine A. The $p$ value results of these analyses are shown in Tables 5.7 and 5.8.

Table 5.7: $p$ value results for the ANOVA analysis of correlation between the adjusted CMRR values and roof condition (binary) at Mine A.

<table>
<thead>
<tr>
<th>Adjusted CMRR evaluated with Roof Stability (binary)</th>
<th>$p$ Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>CMRR (no joint cohesion or roughness parameter)</td>
<td>0.78</td>
</tr>
<tr>
<td>CMRR (no joint spacing or persistence parameter)</td>
<td>0.089</td>
</tr>
<tr>
<td>CMRR (no strength parameter)</td>
<td>0.066</td>
</tr>
<tr>
<td>CMRR (no moisture sensitivity parameter)</td>
<td>0.23</td>
</tr>
</tbody>
</table>

Table 5.8: $p$ value results for the ANOVA analysis of correlation between the adjusted CMRR values and roof stability on a scale of 1-4 at Mine A.

<table>
<thead>
<tr>
<th>Adjusted CMRR evaluated with Roof Stability (1-4 scale)</th>
<th>$p$ Value from ANOVA</th>
<th>$p$ Value from Spearman</th>
</tr>
</thead>
<tbody>
<tr>
<td>CMRR (no joint cohesion or roughness parameter)</td>
<td>0.13</td>
<td>0.022</td>
</tr>
<tr>
<td>CMRR (no joint spacing or persistence parameter)</td>
<td>0.0018</td>
<td>0.018</td>
</tr>
<tr>
<td>CMRR (no strength parameter)</td>
<td>0.0017</td>
<td>0.0045</td>
</tr>
<tr>
<td>CMRR (no moisture sensitivity parameter)</td>
<td>0.0095</td>
<td>0.0065</td>
</tr>
</tbody>
</table>
5.5.2 Mine B

The four primary components of the CMRR (joint cohesion and roughness, joint spacing and persistence, strength and moisture sensitivity were removed) from the original CMRR values and four adjusted CMRR values were calculated for each data collection location. The adjusted CMRR values were analyzed for correlation with roof stability (as a binary parameter and on a scale of 1-4) at Mine B. The \( p \) value results of these analyses are shown in Tables 5.9 and 5.10. The boxplot result for the CMRR with no joint spacing or persistence parameter, evaluated for correlation with roof stability scale is shown in Figure 5.5.

Table 5.9: \( p \) value results for the ANOVA analysis of correlation between the adjusted CMRR values and roof stability (binary) at Mine B.

<table>
<thead>
<tr>
<th>Adjusted CMRR</th>
<th>( p ) Value from ANOVA</th>
</tr>
</thead>
<tbody>
<tr>
<td>CMRR (no joint cohesion or roughness parameter)</td>
<td>0.19</td>
</tr>
<tr>
<td>CMRR (no joint spacing or persistence parameter)</td>
<td>0.18</td>
</tr>
<tr>
<td>CMRR (no strength parameter)</td>
<td>0.22</td>
</tr>
<tr>
<td>CMRR (no moisture sensitivity parameter)</td>
<td>0.27</td>
</tr>
</tbody>
</table>

Table 5.10: \( p \) value results for the ANOVA analysis of correlation between the adjusted CMRR values and roof stability on a scale of 1-4 at Mine B.

<table>
<thead>
<tr>
<th>Adjusted CMRR</th>
<th>( p ) Value from ANOVA</th>
<th>( p ) Value from Spearman</th>
</tr>
</thead>
<tbody>
<tr>
<td>CMRR (no joint cohesion or roughness parameter)</td>
<td>0.83</td>
<td>0.55</td>
</tr>
</tbody>
</table>
Table 5.10: (continued)

<table>
<thead>
<tr>
<th>CRRR (no joint spacing or persistence parameter)</th>
<th>0.039</th>
<th>0.22</th>
</tr>
</thead>
<tbody>
<tr>
<td>CRRR (no strength parameter)</td>
<td>0.90</td>
<td>0.80</td>
</tr>
<tr>
<td>CRRR (no moisture sensitivity parameter)</td>
<td>0.98</td>
<td>0.83</td>
</tr>
</tbody>
</table>

Figure 5.5: Boxplot result for CRRR (no joint spacing or persistence) evaluated for correlation with the roof stability 1-4 scale using the ANOVA analysis.

5.6 Logistic Regression

As discussed in Chapter 4.7, logistic regression analysis was carried out, as it returns a practically useable probability value describing how likely it is a certain combination of the continuous predictor variables will correspond to a given category of roof stability. Since the predictor variables must be continuous and the response variables categorical, logistic regression was used to examine the relationship between the CRRR and roof stability, both as a binary parameter and on a scale of 1-4. The result of logistic regression is a model equation giving the log odds of the response
variable being in one category versus another. This is then converted to a probability of whether the response variable will be in one category versus another. The statistical significance of the model coefficients ($p$ values) were also. In all of the equations presented in this chapter, $X_{CMRR}$ refers to an arbitrary input CMRR value, $X_{SlopeAngle}$ to an arbitrary input slope angle value, etc.

5.6.1 Mine A

The logistic regression model for the analysis of binary roof stability and CMRR at Mine A is shown in Equation 5.1.

\[
\ln \left( \frac{\text{stable}}{\text{unstable}} \right) = -1.57 + 0.075 X_{CMRR} \tag{5.1}
\]

The $p$ values of these coefficients are 0.35 and 0.20 respectively. A plot of probability values (for binary roof stability category) versus CMRR is shown in Figure 5.6. The results of the logistic regression analysis of roof stability on a scale of 1-4 with CMRR are shown in Equations 5.2, 5.3 and 5.4.

\[
\ln \left( \frac{\text{Category 1}}{\text{Category 4}} \right) = 2.6 - 0.21 X_{CMRR} \tag{5.2}
\]

\[
\ln \left( \frac{\text{Category 2}}{\text{Category 4}} \right) = 4.8 - 0.21 X_{CMRR} \tag{5.3}
\]

\[
\ln \left( \frac{\text{Category 3}}{\text{Category 4}} \right) = 8.26 - 0.21 X_{CMRR} \tag{5.4}
\]

The $p$ values of these coefficients are: 0.13 (coefficient = 2.6), 0.0065 (coefficient = 4.8), 0.0002 (coefficient = 8.26) and 0.0016 (coefficient = -0.21). A plot of probabilities for a given CMRR to be in each of the four roof stability categories is shown in Figure 5.7.
Figure 5.6: A plot of probability values with the CMRR as the predictor variable and binary roof stability as the response.
Next the depth of cover parameter was input as the predictor variable and roof stability on a scale of 1-4 was the response. The resulting logistic regression model is presented in Equations 5.5-5.7. The $p$ values for the coefficients are 0.57 (coefficient = -1.3), 0.83 (coefficient = 0.48), 0.2 (coefficient = 2.9) and 0.55 (coefficient = -0.001). A plot of probabilities for each of the four roof stability categories is shown in Figure 5.8.

\[
\text{ln (Category 1/Category 4) = -1.3 – 0.001 X_{Depth of Cover}}
\]  
(5.5)

\[
\text{ln (Category 2/Category 4) = 0.48 – 0.001 X_{Depth of Cover}}
\]  
(5.6)

\[
\text{ln (Category 3/Category 4) = 2.9 – 0.001 X_{Depth of Cover}}
\]  
(5.7)
Slope Angle was evaluated with roof stability scale next; the resulting logistic regression model is presented in Equations 5.8, 5.9 and 5.10. The \( p \) values for the coefficients are 0.002 (coefficient = -2.5), 0.19 (coefficient = -0.74), 0.009 (coefficient = 1.7) and 0.82 (coefficient = -0.012). A plot of probabilities for each of the four roof stability categories is shown in Figure 5.9.

\[
\ln \left( \frac{\text{Category 1}}{\text{Category 4}} \right) = -2.5 - 0.012 \times \text{Slope Angle} \tag{5.8}
\]

\[
\ln \left( \frac{\text{Category 2}}{\text{Category 4}} \right) = -0.74 - 0.012 \times \text{Slope Angle} \tag{5.9}
\]

\[
\ln \left( \frac{\text{Category 3}}{\text{Category 4}} \right) = 1.7 - 0.012 \times \text{Slope Angle} \tag{5.10}
\]
Figure 5.9: A plot of probability values with the slope angle as the predictor variable and the roof stability categories 1-4 as the response.

Next, the depth of cover parameter was combined with the CMRR and together they were input as the predictor variables. The resulting logistic regression model with roof stability (on a 1-4 scale) as the response is shown in Equations 5.11-5.13. The $p$ values of the coefficients are 0.2 (coefficient = 3.6), 0.046 (coefficient = 5.9), 0.0047 (coefficient = 9.3), 0.0017 (coefficient = -0.21) and 0.63 (coefficient = -0.0012). Surface plots of the probabilities for each of the four roof condition categories are shown in Figure 5.10.

\[
\ln \left( \frac{\text{Category 1}}{\text{Category 4}} \right) = 3.6 - 0.21X_{\text{CMRR}} - 0.0012X_{\text{Depth of Cover}}
\]  

(5.11)
\[
\ln (\text{Category 2/Category 4}) = 5.9 - 0.21X_{\text{CMRR}} - 0.0012X_{\text{Depth of Cover}} \quad (5.12)
\]
\[
\ln (\text{Category 3/Category 4}) = 9.3 - 0.21X_{\text{CMRR}} - 0.0012X_{\text{Depth of Cover}} \quad (5.13)
\]

Figure 5.10: Surface plots of probabilities with the CMRR and depth of cover as the predictor variables and the roof condition scale as the response.

The slope angle parameter was then combined with the CMRR as the input predictor variables and evaluated with roof stability (on a 1-4 scale). The resulting logistic regression model is shown in Equations 5.14-5.16. The \( p \) values of the coefficients are 0.13 (coefficient = 2.6), 0.0069 (coefficient = 4.8), 0.0003 (coefficient =
8.3), 0.0017 (coefficient = 0.2) and 0.92 (coefficient = 0.0056) and surface plots of the probabilities for each of the four categories are shown in Figure 5.11.

\[
\ln \left( \frac{\text{Category 1}}{\text{Category 4}} \right) = 2.6 - 0.2X_{CMRR} - 0.0056X_{\text{SlopeAngle}} \\
\ln \left( \frac{\text{Category 2}}{\text{Category 4}} \right) = 4.8 - 0.2X_{CMRR} - 0.0056X_{\text{SlopeAngle}} \\
\ln \left( \frac{\text{Category 3}}{\text{Category 4}} \right) = 8.3 - 0.2X_{CMRR} - 0.0056X_{\text{SlopeAngle}}
\]

(5.14)  
(5.15)  
(5.16)

Figure 5.11: Surface plots of probabilities with the CMRR and slope angle as the predictor variables and roof condition scale as the response.

Lastly the slope angle and the depth of cover were combined with the CMRR as the predictor variables. The resulting logistic regression model with roof condition scale
as the response is shown in Equations 5.17-5.19. The $p$ values of the coefficients are 0.2 (coefficient = 3.7), 0.04 (coefficient = 6.0), 0.005 (coefficient = 9.4), 0.0018 (coefficient = 0.2), 0.0012 (coefficient = 0.63), 0.87 (coefficient = 0.009). The results of this model are not presented visually due to the large number of variables involved.

\[
\ln (\text{Category 1/Category 4}) = 3.7 - 0.2X_{\text{CMRR}} - 0.0012X_{\text{Depth of Cover}} - 0.009X_{\text{Slope Angle}}
\]  
(5.17)

\[
\ln (\text{Category 2/Category 4}) = 6.0 - 0.2X_{\text{CMRR}} - 0.0012X_{\text{Depth of Cover}} - 0.009X_{\text{Slope Angle}}
\]  
(5.18)

\[
\ln (\text{Category 3/Category 4}) = 9.4 - 0.2X_{\text{CMRR}} - 0.0012X_{\text{Depth of Cover}} - 0.009X_{\text{Slope Angle}}
\]  
(5.19)

### 5.6.2 Mine B

The logistic regression model for the analysis of binary roof stability and CMRR at Mine B is shown in Equation 5.20.

\[
\ln (\text{stable/unstable}) = -1.6 + 0.038X_{\text{CMRR}}
\]  
(5.20)

The $p$ values of these coefficients are 0.31 and 0.22 respectively. A plot of probability values for the CMRR to correspond to stable or unstable roof are shown in Figure 5.12.

The results of the logistic regression analysis of roof stability on a scale of 1-4 with CMRR are shown in Equations 5.21, 5.22, and 5.23.

\[
\ln (\text{Category 1/Category 4}) = -1.6 - 0.004X_{\text{CMRR}}
\]  
(5.21)

\[
\ln (\text{Category 2/Category 4}) = 0.2 - 0.004X_{\text{CMRR}}
\]  
(5.22)

\[
\ln (\text{Category 3/Category 4}) = 1.2 - 0.004X_{\text{CMRR}}
\]  
(5.23)
Figure 5.12: A plot of probability values with the CMRR as the predictor variable and binary roof stability as the response.

The $p$ values of all of these coefficients are 0.26 (coefficient = -1.6), 0.88 (coefficient = 0.2), 0.4 (coefficient = 1.2) and 0.88 (coefficient = -0.004). A plot of probabilities for a given CMRR to be in each of the four roof stability categories is shown in Figure 5.13.

Next, the depth of cover was input as the predictor variable with roof stability scale as the response. The resulting logistic regression model is shown in Equations 5.24-5.26 and a plot of probabilities for each of the four roof stability categories is shown.
Figure 5.13: A plot of probability values with the CMRR as the predictor variable and the roof stability categories 1-4 as the response.
in Figure 5.14. The \( p \) values of the coefficients are 0.23 (coefficient = -1.08), 0.33 (coefficient = 0.84), 0.04 (coefficient = 1.9) and 0.3 (coefficient = -0.0017).

\[
\ln (\text{Category 1/Category 4}) = 1.08 - 0.0017X_{\text{Depth of Cover}} \tag{5.24}
\]

\[
\ln (\text{Category 2/Category 4}) = 0.84 - 0.0017X_{\text{Depth of Cover}} \tag{5.25}
\]

\[
\ln (\text{Category 3/Category 4}) = 1.9 - 0.0017X_{\text{Depth of Cover}} \tag{5.26}
\]

Figure 5.14: A plot of probability values with the Depth of Cover as the predictor variable and the roof stability categories 1-4 as the response.
Slope angle was input next as the predictor variable. The resulting logistic regression model with roof stability on a scale of 1-4 as the response is shown in Equations 5.27-5.29. The \( p \) values of the coefficients are 0.0 (coefficient = -3.7), 0.04 (coefficient = -1.5), 0.68 (coefficient = -0.28) and 0.017 (coefficient = 0.12). A plot of the probability values for each roof stability value with slope angle is presented in Figure 5.15.

\[
\ln \left( \frac{\text{Category 1}}{\text{Category 4}} \right) = 1.08 - 0.0017X_{\text{Depth of Cover}} \quad (5.27)
\]

\[
\ln \left( \frac{\text{Category 2}}{\text{Category 4}} \right) = 0.84 - 0.0017X_{\text{Depth of Cover}} \quad (5.28)
\]

\[
\ln \left( \frac{\text{Category 3}}{\text{Category 4}} \right) = 1.9 - 0.0017X_{\text{Depth of Cover}} \quad (5.29)
\]

The third continuous non-CMRR parameter examined at mine B was topographic curvature. This parameter was input as the predictor variable and roof stability scale was the response. The resulting logistic regression model is shown in Equations 5.30-5.32 and a plot of the probabilities for each roof stability category is presented in Figure 5.16. The \( p \) values for the coefficients are 0.001 (coefficient = -2.3), 0.4 (coefficient = -0.42), 0.23 (coefficient = 0.63) and 0.29 (coefficient = 1027).

\[
\ln \left( \frac{\text{Category 1}}{\text{Category 4}} \right) = -2.3 - 1027X_{\text{Topographic Curvature}} \quad (5.30)
\]

\[
\ln \left( \frac{\text{Category 2}}{\text{Category 4}} \right) = -0.42 - 1027X_{\text{Topographic Curvature}} \quad (5.31)
\]

\[
\ln \left( \frac{\text{Category 3}}{\text{Category 4}} \right) = 0.63 - 1027X_{\text{Topographic Curvature}} \quad (5.32)
\]

Since topographic curvature and slope angle are shown to have more of an influence on roof stability category than the CMRR, these two variables were combined for a further logistic regression analysis with roof condition scale from 1-4 as the response. The resulting logistic regression model is shown in Equations 5.33-5.35.
Figure 5.15: A plot of probability values with slope angle as the predictor variable and the roof stability categories 1-4 as the response.
Figure 5.16: A plot of probability values with topographic curvature as the predictor variable and the roof stability categories 1-4 as the response.

The $p$ values of the coefficients are 0.0 (coefficient = -4.7), 0.009 (coefficient = -2.3), 0.2 (coefficient = -1.05), 0.13 (coefficient = 1585) and 0.009 (coefficient = 0.14). Surface plots of the probabilities for each of the four roof condition categories are shown in Figure 5.17.

\begin{align*}
\ln(\text{Category 1}/\text{Category 4}) &= -4.7 - 0.14X_{\text{Slope Angle}} + 1585X_{\text{Topographic Curvature}} \\
\ln(\text{Category 2}/\text{Category 4}) &= -2.3 - 0.14X_{\text{Slope Angle}} + 1585X_{\text{Topographic Curvature}} \\
\ln(\text{Category 3}/\text{Category 4}) &= -1.05 - 0.14X_{\text{Slope Angle}} + 1585X_{\text{Topographic Curvature}}
\end{align*}
The slope angle was next combined with the depth of cover as inputs to the logistic regression analysis with roof condition scale from 1-4 as the response. The resulting logistic regression model is shown in Equations 5.36-5.38. The \( p \) values of the coefficients are 0.013 (coefficient = -2.9), 0.54 (coefficient = -0.6), 0.55 (coefficient = 0.6), 0.015 (coefficient = 0.12) and 0.28 (coefficient = -0.0012). Surface plots of the probabilities for each of the four roof condition categories are shown in Figure 5.18.

\[
\ln(\text{Category 1}/\text{Category 4}) = -2.9 - 0.12\times\text{SlopeAngle} - 0.0012\times\text{DepthofCover} \quad (5.36)
\]

\[
\ln(\text{Category 2}/\text{Category 4}) = -0.6 - 0.12\times\text{SlopeAngle} - 0.0012\times\text{DepthofCover} \quad (5.37)
\]

\[
\ln(\text{Category 3}/\text{Category 4}) = 0.6 - 0.12\times\text{SlopeAngle} - 0.0012\times\text{DepthofCover} \quad (5.38)
\]

The topographic curvature at Mine B was then combined with the depth of cover for a further logistic regression analysis again with roof stability on a scale of 1-4 as the response. The resulting logistic regression model is shown in Equations 5.39-5.41. The \( p \) values of the coefficients are 0.15 (coefficient = -1.7), 0.8 (coefficient = 0.27), 0.24 (coefficient = 1.3), 0.49 (coefficient = -0.0012) and 0.45 (coefficient = 779). Surface plots of the probabilities for each of the four roof condition categories are shown in Figure 5.19.

\[
\ln(\text{Category 1}/\text{Category 4}) = -1.7 - 0.0012\times\text{DepthofCover} + 779\times\text{TopographicCurvature} \quad (5.39)
\]

\[
\ln(\text{Category 2}/\text{Category 4}) = 0.27 - 0.0012\times\text{SlopeAngle} + 779\times\text{TopographicCurvature} \quad (5.40)
\]

\[
\ln(\text{Category 3}/\text{Category 4}) = 1.3 - 0.0012\times\text{SlopeAngle} + 779\times\text{TopographicCurvature} \quad (5.41)
\]

Lastly the slope angle was combined with the CMRR, the depth of cover and the topographic curvature as the predictor variables. The resulting logistic regression model with roof condition scale as the response is shown in Equations 5.42-5.44.
Figure 5.17: Surface plots of probabilities with the slope angle and topographic curvature as the predictor variables and roof condition scale as the response.
Figure 5.18: Surface plots of probabilities with the slope angle and depth of cover as the predictor variables and roof condition scale as the response.
The \( p \) values of the coefficients are 0.08 (coefficient = -4.7), 0.38 (coefficient = -2.3), 0.71 (coefficient = -1.0), 0.83 (coefficient = 0.0), 0.66 (coefficient = 0.0), 0.0089 (coefficient = 0.1) and 0.22 (coefficient = 1505). The results of this analysis are not presented visually due to the large number of variables involved.

\[
\ln(\text{Category 1/Category 4}) = -4.7 - 0.0X_{\text{CMRR}} - 0.0X_{\text{Depth of Cover}} - 0.1X_{\text{Slope Angle}} \\
+ 1505X_{\text{Topographic Curvature}} \tag{5.42}
\]

\[
\ln(\text{Category 2/Category 4}) = -2.3 - 0.0X_{\text{CMRR}} - 0.0X_{\text{Depth of Cover}} - 0.1X_{\text{Slope Angle}} \\
+ 1505X_{\text{Topographic Curvature}} \tag{5.43}
\]
ln(Category 3/Category 4) = --1.0 – 0.0X_{CMRR} – 0.0X_{Depth of Cover} – 0.1X_{Slope Angle} 
\hspace{1cm} + 1505X_{TopographicCurvature} \hspace{1cm} (5.44)

5.7 Additional Analyses

5.7.1 Location at a Valley Base

As discussed in Chapter 2.11.7, Griffith et al. (2014) and Pan et al. (1995) noted that the horizontal compressive strength was magnified at valley bases. The location at a valley base was therefore included as a binary (yes/no) parameter and analyzed for correlation with binary roof stability (Fisher Exact Test) and roof stability on a scale on 1-4 (Chi Squared Test).

At Mine A, the $p$ value with binary roof stability was 0.95 and with roof stability scale, 0.21 (8 yes, 22 no). At Mine B, the $p$ values were 0.71 and 0.33 for binary and scale roof stability respectively (6 yes, 24 no).

5.7.2 Loading Condition

The results upon examination of the loading type at each of the data collection locations are shown in Tables 5.11 and 5.12.

Table 5.11: Summary of Loading Type and Corresponding Roof Stability Classification at Mine A.

<table>
<thead>
<tr>
<th>Loading Type</th>
<th>No. of Locations</th>
<th>Roof Stability Classification (1-4 scale)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Development Loading</td>
<td>6</td>
<td>Category 1 = 1, Category 2 = 2,</td>
</tr>
</tbody>
</table>
<pre><code>                          |                  | Category 3 = 1, Category 4 = 2          |
</code></pre>
Table 5.11: (continued)

<table>
<thead>
<tr>
<th>Loading Type</th>
<th>No. of Locations</th>
<th>Corresponding Roof Stability Classification</th>
</tr>
</thead>
<tbody>
<tr>
<td>Headgate Loading</td>
<td>2</td>
<td>Category 2 = 1, Category 3 = 1</td>
</tr>
<tr>
<td>Recovery Room Loading</td>
<td>22</td>
<td>Category 1 = 1, Category 2 = 4, Category 3 = 14, Category 4 = 3</td>
</tr>
</tbody>
</table>

Table 5.12: Summary of Loading Type and Corresponding Roof Stability Classification at Mine B.

<table>
<thead>
<tr>
<th>Loading Type</th>
<th>No. of Locations</th>
<th>Roof Stability Classification (1-4 scale)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Development Loading</td>
<td>23</td>
<td>Category 1 = 4, Category 2 = 8, Category 3 = 5, Category 4 = 6</td>
</tr>
<tr>
<td>Headgate Loading</td>
<td>7</td>
<td>Category 1 = 0, Category 2 = 3, Category 3 = 2, Category 4 = 2</td>
</tr>
</tbody>
</table>

5.7.3 Support Categories and Roof Stability

At Mine B, the results of evaluating correlation between CMRR and the support categories (as a proxy for roof stability) gave a $p$ value of 0.34. The $p$ values for the correlation strength between the support categories and the non-CMRR parameters collected at Mine B are shown in Table 5.13.

Table 5.13: Summary of $p$ values for the analysis of correlation between support category and the non-CMRR parameters collected at Mine B.

<table>
<thead>
<tr>
<th>Parameter Evaluated with Support Category</th>
<th>$p$ Value from ANOVA</th>
<th>$p$ Value from Chi Squared Test</th>
<th>Number of Data Points Evaluated</th>
</tr>
</thead>
<tbody>
<tr>
<td>Coal Seam Merge</td>
<td>--</td>
<td>0.084</td>
<td>Yes = 6, No = 24</td>
</tr>
<tr>
<td>Rib/Floor Condition</td>
<td>--</td>
<td>0.096</td>
<td>Unstable = 7, Stable = 23</td>
</tr>
<tr>
<td>Intersections</td>
<td>--</td>
<td>0.10</td>
<td>Yes = 17, No = 13</td>
</tr>
</tbody>
</table>
Table 5.13: (continued)

<p>| | | | |</p>
<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Depth of Cover</strong></td>
<td>1e-9</td>
<td>--</td>
<td>30</td>
</tr>
<tr>
<td><strong>Slope of Surface</strong></td>
<td>0.70</td>
<td>--</td>
<td>30</td>
</tr>
<tr>
<td><strong>Topography</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
CHAPTER 6
DISCUSSION

6.1 Roof Stability Classification

Before beginning to discuss the results presented above in the context of the hypothesis, it is necessary to fully explain why and how the roof stability parameter recorded at the two case study mines was recorded as it was.

As mentioned in Chapters 4 and 5, the roof stability was initially recorded as a binary parameter – the roof was stable or unstable. However, in reality, roof stability is a continuum. For example, mine roof can be very stable, marginally stable, marginally unstable, or very unstable. It was therefore decided to test a roof stability scale. From the level of detail in the data recorded, a 1-5 scale was considered first. However, the question of how this related to stable or unstable was not answered satisfactorily. In theory if 5 reflects excellent roof and 1 reflects very poor roof, 1 and 2 would represent unstable roof and 4 and 5, stable roof, however category three would then not correspond to either. This made it more challenging to assign the third category to the data recorded in the field. Detailed criteria corresponding to each category were also required. Having an odd number of categories and thus a middle category is also considered to be less accurate and allow whomever is classifying the data to be less discriminating (Garland, 1991; Losby and Wetmore, 2012).

A scale of 1-4 was considered next, it has the same advantage of being a more continuous scale than the binary classification but fits better with the data recorded. It also removes the inaccuracies associated with having a neutral middle category (Losby
and Wetmore, 2012). The criteria for each of the four categories are shown in Chapter 4.6.2. In theory, a scale with any range could have been chosen, however the number of categories depends on the level of detail in the data recorded. It would also be unnecessary to have a different classification for each site (i.e. 30 locations) as this would essentially mean the parameter is not being classified at all. A scale of 1-4 was therefore used to classify the roof stability, although the results from the main analyses are also presented with the roof classification as binary for comparison.

6.2 Mine A

It should be reiterated here that due the small size of the data set used in these analyses, the relative $p$ values are particularly important, as their absolute values depend on the size of the data set. If a series of values are all above the significance threshold, but one is notably lower than the others, the potential that that factor may be important should not be discounted entirely.

As shown in Figure 5.2, the CMRR appears to be well correlated with roof stability (when classified on a scale of 1-4) at Mine A ($p$ value = 0.0073). However, this is a good example of a case where there is strong statistical significance and reasonable but not perfect predictive capability, as shown in the logistic regression results in Figure 5.7. From Figure 5.7, it can be seen that for a CMRR value greater than 40, the probability of the roof stability falling in Category 1 or 2 is very low. For a CMRR value of 30 there is a high probability the roof stability will be in Category 3. For CMRR values less than 30 the likelihood of them corresponding with roof stability
Category 4 is unlikely, however the other categories are less delineated at these CMRR values.

When the CMRR was evaluated with binary roof stability the p values are much higher, \( p = 0.20 \) (see Figure 5.3). This is thought to be due to the fact that the binary scale is less accurate as it requires more approximation and has less room for variation in stability. The logistic regression results with binary roof stability again indicate reasonable predictive capability (Figure 5.6) with greater predictive confidence for higher CMRR values. Given the greater accuracy of the scale classification of roof stability, the logistic regression results presented in Figure 5.7 are recommended to be used in future work rather than those using the binary classification (Figure 5.6).

It should also be noted here that when the roof stability classification was changed from binary to a 1-4 scale, two locations originally recorded as having unstable roof were reclassified into Category 3 (which generally corresponds to stable roof). The first location was one of the initial measurements; relative to the previous measurement locations it was less stable. Since there were only two options, it was classified as unstable. The second location showed minor movement across a fault (2-3 inches of displacement), however the mine engineers considered no extra support was required and there had been no recent movement. Since there had been minor movement in the past, the location was considered less than stable, however upon re-evaluation of the data collected and comparison with the stability scale criteria, it was classified as Category 3.
Both of these cases highlight the benefits of using the scale classification. Once a quantifiable measure (skin failure) is used along with consistent, specific criteria, there is less ambiguity in how the roof stability is classified.

These results show that the CMRR has a statistically significant relationship with the roof stability scale, but the practical significance of the CMRR (predictive capability) at Mine A is not comprehensive. This would therefore suggest there are additional factors indicative of roof instability at Mine A in addition to those included in the CMRR, despite the statistically significant correlation.

The data presented in Table 5.1 show the presence of faulting or location at an intersection are correlated with the CMRR value at Mine A. The depth of cover, gradient of surface topography and presence of the sandstone channel in the immediate roof are not. The above results make sense; the CMRR does not account for the depth of cover or surface topography (or any aspect of/proxy for stress). The CMRR also does not account for the presence of large scale geological features such as sandstone channels, although this was explicitly stated by Mark and Molinda (1994) when the CMRR was developed.

The CMRR does not explicitly account for roof being weaker at an intersection either. However, the roof at intersections may end up being recorded as having a lower strength, cohesion and friction angle than the surrounding roof due to the increased load from the wider span weakening the rockmass. If this is the case, the CMRR would implicitly capture this parameter. The presence of slickensides is included in the CMRR, but an explicit record of fault displacement or slip is not. However, since the faulting at
Mine A is on a local scale, primarily as slickensides, it seems to be well captured by the CMRR at this mine.

It should be noted, however, that the more important question is whether these non-CMRR parameters are correlated with roof stability. Evaluating their correlation with CMRR is nonetheless important though, as it gives an idea of whether any of these parameters are captured albeit indirectly by the rating system as it stands. Evaluating the correlation between the non-CMRR parameters collected at Mine A and the 1-4 roof stability scale shows faulting to be very strongly correlated with roof stability at Mine A (p=0.014). The depth of cover and slope of surface topography also have relatively low p values (0.08 and 0.07, respectively). However, the logistic regression results (Figures 5.7-5.9) suggest the CMRR has a greater influence on the resulting probabilities than depth of cover or slope angle in terms of predicting roof stability. This suggests that faulting is the most influential parameter in this analysis at this mine. Depth of cover and surface topography appear have a minor effect on roof stability at Mine A, with depth of cover more influential than slope angle (see Figures 5.8-5.11). If a correlation exists, it appears to be overshadowed by the high incidence of faulting at the mine however. The Spearman test results also show faulting to correlate well with binary roof stability and roof stability on a 1-4 scale and topographic slope returns a relatively low p value (0.16).

When the non-CMRR parameters collected are evaluated with binary roof stability at Mine A using the ANOVA analysis, the results are inconclusive. This again could be because the binary categorization approach requires more approximation than a scale and allows for less variation. The results with binary roof stability do not show location at an intersection or the presence of faulting to be statistically correlated with
roof stability, despite the fact that these parameters are well correlated with the CMRR. The depth of cover and surface topography also show little correlation with the binary roof stability; this could be because the geology is the primary control on roof stability at Mine A, overshadowing any effect of stress-related parameters.

Removing each of the four constituent parameters of the CMRR and evaluating their correlation with roof stability on a scale of 1-4 using the ANOVA analysis shows the CMRR with the discontinuity cohesion and roughness parameter removed to give a much higher \( p \) value (0.13) compared to the original CMRR (0.0073). The Spearman tests also showed the removal of the cohesion and roughness parameter to give a higher value compared to the removal of the other parameters (see Table 5.8). These results imply that the discontinuity cohesion and roughness parameter is particularly influential at this mine, and much more so than the other CMRR parameters that, when removed, the resulting \( p \) values were relatively unchanged from the originals. This is consistent with the observation that small scale faulting and slickensides are prevalent at Mine A.

### 6.3 Mine B

At Mine B, the CMRR does not appear to correlate with the roof stability, giving \( p \) values of 0.95 and 0.24, respectively, for the scale and binary roof stability analyses. In particular, this would imply there are other factors in addition to those included in the CMRR which are controlling roof instability at Mine B and that the CMRR does not fully explain roof instability at this mine. The logistic regression results evaluating CMRR with roof stability on a 1-4 scale also support this conclusion (as shown in Figure 5.13) as a
CMRR of any value has approximately the same probability of falling into a given roof stability category. The logistic regression analysis for the CMRR with binary roof condition (Figure 5.12) shows the CMRR to have reasonable predictive capacity especially at higher values. However, as discussed previously, the binary classification of roof condition is thought to be less representative of roof stability than the 1 – 4 scale, so these results should be treated with caution.

When evaluated with the non-CMRR parameters collected at Mine B (using the ANOVA analysis), the CMRR shows a statistically significant correlation with the coal seam merge \((p=0.0077)\) and topographic curvature. All other \(p\) values from this analysis are much higher (see Table 5.2). The topographic curvature may correlate with the CMRR due to the stress rotation from slope changes causing the roof to be weaker at a given location; this would then translate into a lower CMRR value particularly if the changes in loading translated to a higher incidence of discontinuities or weakened the existing discontinuities. This is speculation however, and cannot be proven here. Although the merge shows a strong correlation it is in the opposite direction to that expected – when the merge was recorded as present, the CMRR was consistently high. This is thought to be because the interburden between the coal seams is stronger and had a consistently higher CMRR value than the material above the rider seam. Therefore, when the rider seam was in the roof or above the bolted interval but still within 12 ft of the roof, a significant proportion of the interburden was in the bolted interval. The ratio of interburden to rider coal could have been sufficiently high that the lower CMRR of the rider coal was overshadowed by the stronger interburden. It is possible that the rider coal has a negative impact on stability even when it is above the
bolted interval implying the interburden-coal contact is so weak that even relatively far above the main seam it can still have a negative impact on stability.

When roof stability on a scale of 1-4 was evaluated with the non-CMRR parameters collected it was found that location at an intersection showed a statistically significant correlation (ANOVA $p = 0.054$ and Spearman $p = 0.0079$). Rib/Floor condition also showed statistical significance in the Chi Squared test (Table 5.6). The depth of cover parameter returned a low $p$ value (0.024) for the ANOVA analysis but a high value in the Spearman test. This is likely due to the input data not exactly fitting with that required by the Spearman test. The logistic regression results (Figure 5.14) also show depth of cover to have more of an impact on the resulting probability values compared to the CMRR. The correlation with binary roof stability also shows intersection location to be significant but it is slope angle and topographic curvature which are the other significant non-CMRR parameters. This is supported by the logistic regression results (Figures 5.15 and 5.16) which show slope angle to have a greater influence on the probability of roof stability than the CMRR and depth of cover. The topographic curvature is also shown to be influential although arguably less so than the slope angle. It should be noted that the notably higher coefficient values for topographic curvature in Equations 5.30-5.35 and 5.39-5.44 are likely due to the small absolute values of topographic curvature that were calculated. Nonetheless, topographic curvature seems to be an influential parameter on roof stability at Mine B. To further examine the interaction between slope angle and topographic curvature at Mine B they were jointly input as predictor variables in the logistic regression analysis. The results in Figure 5.17 suggest that slope angle has more influence on roof stability
at Mine B compared to topographic curvature and also than the relationship between
the two is non-linear. Where the slope angle is high and the topographic curvature is
low, the probability of the roof being unstable (in Category 1) is high. This might suggest
that a consistently steep slope is most problematic for roof stability.

The coal seam merge again shows a potentially spurious correlation as
whenever it was recorded as present the roof stability was consistently good. Faulting
was not recorded anywhere at Mine B as there was no evidence of it. The removal of
the component parameters of the CMRR shows the joint spacing and persistence
parameter to be the least influential at Mine B – when removed, the \( p \) value showing
correlation with roof stability scale drops significantly to 0.039 (ANOVA analysis). When
the other parameters are removed, the \( p \) values remain very close to the original value.
The main discontinuities at Mine B are bedding planes, which are relatively persistent,
however the sandstone in the roof is generally massive meaning discontinuity
characteristics are less relevant. This is a possible reason for the aforementioned result.
Figure 5.5 also shows that Category 2 (which has the greatest number of data points)
correlates with high CMRR values, however Categories 1 and 2 were developed to
correspond with unstable roof. This suggests the low \( p \) value could be spurious as
unstable roof doesn’t correspond to low CMRR values despite the statistically significant
\( p \) value.

As with the curvature of surface topography, the analyses using support category
as a proxy for roof stability were only carried out at Mine B due to the available data.
The coal seam merge and depth of cover both give low \( p \) values, however given that the
support criteria are based on the depth of cover, slope of surface topography and the
coal seam merge, this is unsurprising. It is perhaps more unexpected that the slope of topography did not return a low \( p \) value as well, although it could be that not enough of the data collection locations were under areas of with steep slopes. It is also possible that the mine engineers are mostly ignoring topographic slope in their decisions, although this seems unlikely given that slope angle is explicitly included in the support categories.

### 6.4 Comparative Results

Based on the previously discussed results, it appears that there are additional factors indicative of roof instability at both mines which are not included in the CMRR. It does however seem that the CMRR is more effective at Mine A and has greater predictive capability there than at Mine B.

Location at an intersection shows a statistically significant correlation with roof stability at Mine B. At Mine A, the CMRR and intersection location correlate well, but not at Mine B. This suggests intersection location is a parameter which merits further investigation and is one which could potentially be added to the CMRR with significantly more data to support it. At Mine A, faulting shows a strong correlation with CMRR and with roof stability, however this cannot be compared to Mine B as no faulting was observed there. The stress related parameters such as depth of cover and surface topography show a strong relationship with roof stability at Mine B, but less so at Mine A.

The analyses involving the removal of the CMRR component parameters, also highlights how variable conditions can be between mines and how challenging it is for a
classification system to satisfactorily capture the conditions at multiple mines; the most important parameter at one mine can be the least important at another. It is possible that the CMRR has the correct core parameters included in it but they have the wrong rating or ranking system, although this may be very difficult to say as it likely varies between mines.

Location at a valley base returned relatively low $p$ values when examined for correlation with the roof condition scale. The potential significance of these results cannot be discounted as relatively few data points were collected beneath valley bases (8 at Mine A, 6 at Mine B) and the data set as a whole is relatively small. This effect of location beneath a valley base on the CMRR and roof stability is something which could be explored further in future work with a larger and more focused data set. The examination of loading type did not return any conclusive results. This is likely because one type of loading dominated at each mine, meaning any patterns or correlations with roof stability which may exist were not observed.

6.5 Assumptions, Uncertainties and Limitations

As discussed in Chapter 2.10, the inherent limitations of the CMRR impact the interpretations that can be made from the results obtained. As the only one collecting the data underground and calculating the CMRR, the author was as consistent as possible within the data set. However, it remains the case that several people calculating the CMRR could come up with several different answers. Unfortunately, the data set used here (60 data points) is small enough that a small change in the input i.e. reclassifying one or two cases can have a notable effect on the results. It is also
possible that there is insufficient data or a bias in the data collection locations that resulted in data which is not fully representative of Mines A or B.

Using a scale for roof stability certainly reduces the potential for error, with measurable requirements for each category, however there may still be some error where the classification is on the edge of two categories. The rib and floor condition parameter was classified as stable or unstable in the same way the roof stability originally was, although this parameter may be best classified on a continuum too. Given the notable difference in outcome between using binary roof stability and a scale, the rib and floor condition could change significantly if it were to be recorded on a scale too. This is something that could be included in future work.

There is also the potential for error in the methods used to calculate slope angle and topographic curvature. Any method requiring inference between the contours accrues resolution error. This combined with relatively large-scale maps and the human error from carrying out the measurements by hand means these results are less accurate than would be ideal.

It should also be noted that although horizontal stress has been found to be highly influential on roof stability (Griffith et al., 2014), it was not directly measured underground at either case study mine or evaluated in the context of the CMRR in this study. This was partly because of the challenges involved with the accurate measurement of horizontal stress underground and also because depth of cover and topographic relief can be considered as proxies for this parameter. The potential for explicit inclusion of horizontal stress in the CMRR is out with the scope of this study.
The different loading conditions (e.g. headgate loading or development loading) at each data collection site were also not considered in the fieldwork component of this work. Loading condition was assumed to be constant throughout the two mines when the data was collected. However, this is not the case in reality; for this reason, a cursory examination of the data was made with respect to loading condition. This was inconclusive likely due to one loading type dominating at each mine. The effect of loading condition and its relationship with roof stability and the CMRR is something which could be further explored in future work. To do this, a focused data set would be required to get a representative sample of the different loading types. The difference in stress condition as a result of the loading would certainly contribute to explaining why areas even with consistent geology can have notably different roof stabilities. Not scaling the data with respect to loading condition is certainly a source of error and likely has an influence on the results although the significance of this influence is not known.

The effect of time on the CMRR values was not accounted for in this work either but is also something which could be included in future work. For example, the stability of the roof could be very different immediately when support was installed compared to several years after it was first exposed. Ideally the data would be scaled to account for this, making the comparison between sites more realistic.

As mentioned in Chapter 4.6.2, the record of roof stability at the case study mines was also likely affected by the fact that all of the data collection areas were previously supported. The fact that the roof stability scales focus on skin failure should mitigate this by comparing each of the collection sites on the same level. Nonetheless, it should be considered that one mine could be over supporting their roof relative to the
other, meaning there is much less evidence of instability at one versus the other, thereby skewing the results. Perhaps more importantly however, since all of the quantitative analyses were done within one mine, not between mines is how the adjustment of support within one mine might affect the results. For example, if Mine B changed from 1 m bolt spacing to 2 m bolt spacing there would be a lot less potential for skin failure in the latter case, even if the geology were identical.

A significant limit to a study such as this is also a relative lack of breadth of investigation. For the purposes of this study, two case study mines were sufficient to start investigating where the CMRR may be lacking and how or why it may be less applicable in the Western US. However, a much larger data set, incorporating tens to hundreds of mines would be needed to support any definitive recommendations with respect to changing or updating the existing CMRR system. This study also only considers longwall mines; a larger data set would therefore benefit from including different underground mining methods, allowing for a more comprehensive analysis of the CMRR.

6.6 Differences Between Mine A and Mine B

Given the notably different geological histories of the mines, it seems likely that this is a major factor in their difference. As discussed in Chapter 3, the region mined by Mine A was formed due to the Laramide orogeny and has an extensive and complex structural history. Mine B however is located in a broad, gentle synclinal basin. Large-scale folding is therefore present at Mine B, but faulting is very minor or not present. From the review of site-specific literature presented in Chapter 3, it seems that
Mine A is in a foreland basin (developed adjacent and parallel to a mountain belt) but Mine B is located in an intracratonic basin (within stable continental crust). Mine A therefore has significantly more abundant discontinuities and a high incidence of localized faulting and slickensides. The CMRR focus on discontinuities captures this – the cohesion and roughness of a slickenside is included, as is the discontinuity spacing and persistence in a highly faulted area. Mine B however, has no evidence of faulting, with the primary discontinuities present being the bedding in the rock. Much of the sandstone observed in the immediate roof was massive. However, simply having fewer or less persistent discontinuities should not mean the CMRR would not apply; it should, however, suggest the CMRR values would be higher at Mine B (which they are).

Since there is moisture sensitive rock present in the roof strata at both mines, this parameter should be relevant at both mines. Mine A has a higher incidence of mudstone in the immediate roof compared to Mine B, but this should simply be reflected in the CMRR values. With that in mind, the presence of moisture sensitive rock does not seem to be a significant source of difference between mines or explain why the CMRR is more effective at one but not the other.

The shales at Mine B are generally stronger than those at Mine A (2600-4600 psi and 190-800 psi, respectively). Sandstone strengths at Mine B are lower than at Mine A (2300-4800 psi and 4300-7200 psi, respectively). The siltstone present at Mine B was found to have a UCS value of 4300 psi while those present at Mine B presented a much greater variation (740-7400 psi). The coal strengths at both mines are very similar (approx. 800 psi). (Maleki and Gulliver, 1988, Burkhard and Herth, 2016). These variations certainly indicate variation between the two mines, but again shouldn’t be
expected to cause the CMRR to be less effective at one mine compared to the other, as different strengths should simply mean the CMRR values would be different.

The more complex geological history at Mine A also meant additional features such as the paleo sandstone channels are present there, although they were not found to have a significant impact on the roof stability at the mine (none of the $p$ values evaluating correlation between roof stability and sandstone channel presences were close to the threshold for statistical significance of 0.05).

What does seem possible however, is that when the incidence and severity of discontinuities are less (as at Mine B), the stress related parameters become more influential and the CMRR’s focus on geological features becomes less relevant. The results in Table 5.4 (ANOVA Analysis) show depth of cover and slope of surface topography to be influential at Mine A (the relative $p$ values are low). Faulting is also strongly correlated with roof stability at Mine A and the CMRR shows greater predictive capability there (along with a statistical correlation between the CMRR and roof stability). At Mine B, the depth of cover, slope of topography and topographic curvature are related to roof stability (see Table 5.5 and Figures 5.14-5.16), but there is no faulting and fewer discontinuities, and the CMRR has little predictive power. There is no statistically significant relationship with roof stability at Mine B either. These results combined suggest the CMRR might be less capable of predicting roof stability at Mine A if there was no faulting present there. To fully test the hypothesis that the CMRR is less effective in areas with less complex geological structure would require a much larger data set spanning several more mines.
6.7 Differences Between Mines A and B and the Mines in the CMRR Reference Database

To more broadly examine whether the CMRR is more effective at Mine A due to the mines’ more complex geological structure, a closer examination of the mines in the CMRR reference database (Molinda and Mark, 1994) was made. If Mine A is in a similar geologic setting to the mines in the database, this could explain why the CMRR is more effective there; the geological conditions at Mine A are similar to those which were considered in the CMRR development process. If Mine B matches the mines in the database i.e. if it seems the CMRR was developed for the geological conditions at Mine B, yet this study has shown the CMRR to have little predictive capability there, it could be that Mine B is geologically anomalous.

As mentioned in Chapter 2.7, only 11 out of the 96 exposures (at 75 mines) in the data set collected during the CMRR development process were from the Western US: four from Colorado, five from Utah and one each from Wyoming and New Mexico. The CMRR reference database was published as an appendix to the original publication by Molinda and Mark (1994). The names of the mines were withheld, but information on the coal seam mined was not. Where information on the stratigraphy and geology of the area was available, further information on the setting of the sites used as case studies was obtained. This then allows a comparison between the case study mines and those in the CMRR reference database to be made.
6.7.1 Western US Mines in the CMRR Reference Database

Since both Mine A and Mine B are located in the Western US, it is first necessary to examine the similarities and differences between them and the Western mines in the CMRR reference database. Should Mine A match the Western mines in the database but not Mine B, it could be that the small sample size of Western mines meant the geologic conditions at Mine B were simply not considered when the CMRR was developed.

The mines in the CMRR reference database from Colorado mine two different seams. The first seam is located in the same formation as Mine B. The region containing this formation is a large irregular syncline or structural basin with a nearly N-S axis (Campbell, 1923). The surrounding stratigraphy is primarily sandstone which is commonly massive. The regional basin features notable westward facing dips in the sandstone as well as three large faults in the basin and multiple minor localized faults (Campbell, 1923). The area encompassing the second seam forms part of an extensive structural basin with the coal bearing rocks inclining gently towards the axis of the syncline. A later laccolithic intrusion into the sedimentary rocks surrounding the coal beds led to the beds being inclined in various directions (Lee, 1912).

The New Mexico mine in the database mined the York Canyon coal seam in the Raton Coalfield. The basin in which this coal field is located is a broad, arcuate depositional trough and is considered to be a ‘sister’ basin to the basin containing Mine A which contains older but related rock (Pillmore 1991). The axis of the Raton basin separates steeply dipping strata in the West from gently dipping beds in the East. Coal offsets and localized faulting with small displacements are common. An anticline with
approximately 2500 ft of relief is the only major structure in the coal field. The stratigraphy is broadly similar to the basin in which Mine A is located with a sandstone unit underlying the coal-bearing formation. Interbedded in the coal-bearing formation are sandstone, siltstone, shale, carbonaceous mudstone and mudstone. Igneous intrusions crop out in many places in the Raton coalfield (Pillmore, 1991).

The Wyoming coal seam was in the Hanna coal field. Limited information on this field was obtained, however both folding and faulting appear to be present within the Hanna Basin, particularly at its edges (Glass and Roberts, 1980).

The last Western state included in the CMRR reference database was Utah. All five of the seams included were in the Wasatch Plateau coal field. The Wasatch plateau lies between the domelike uplift of the San Rafael Swell and the Great basin to the West. The Great Basin is highly faulted and structurally complex. Because it lies in the transition between these two areas, the Wasatch plateau contains some of both structures. Faulting is prevalent throughout the coal field and has been known to cause problems when mining (Kirshbaum and Biewick, 2000). The strata in the area is slightly tilted to essentially flat. However, localized disturbances cause the dip in some areas to be between 10 and 20 degrees (Kirshbaum and Biewick, 2000).

The presence of faulting is common to all of these locations. All of the Utah data comes from a structurally complex area with significant faulting, similar to the geological conditions at Mine A. Indeed, the mine in New Mexico is located in a ‘sister’ basin to Mine A containing older but related rocks (Pillmore, 1991). It therefore makes sense why the CMRR is effective at Mine A; the geologic conditions there are similar to those considered when the CMRR was developed. Mine B is located in a broad, gentle
synclinal basin, which would be most similar to the setting of the D seam exposure in Colorado. If this is the case, only one of the 97 exposures reasonably matches the environment at Mine B. It is also possible that the mine exposure in Wyoming more closely matches the conditions at Mine B, although that cannot be concluded based on the available information. The CMRR could therefore be less effective at Mine B simply because the geological conditions there were not considered when the CMRR was developed. To further see if this is the case, an examination of the mines in the Eastern US was made.

6.7.2 Eastern US Mines in the CMRR Reference Database

The geological conditions of the Eastern US mines in the CMRR reference database were examined next. Should Mine A closely match the reference mines across the whole country, this would suggest that the CMRR is effective at Mine A because the geological conditions are similar to those for which the CMRR was developed. If there are Eastern mines with similar geology to Mine B in the database, it may be more likely that Mine B is a geologic anomaly. If Mine B does not closely match any mines in the database, it is still possible the mine is in an anomalous geological setting but it could also be that the CMRR database doesn't fully represent the varied geological conditions at mines throughout the US.

Of the 96 coal seams and the corresponding CMRR measurements in the CMRR reference database, 85 were in the Eastern states; most of the sites were located in Illinois, Kentucky, Ohio, Pennsylvania and West Virginia. Many measurements came
from the Pittsburgh and Coalburg coal seams in the Appalachian Basin, with nearly as many coming from the Herrin coal seam in the Illinois Basin.

The Appalachian basin has seen multiple mountain building events leading to the exposure of folded and thrust-faulted rocks. The NW edge of the basin is a broad homocline (where the layers of rock strata dip uniformly in the same direction). The basin itself is structurally complex as with many orogenic belts, with much evidence of folding and faulting and the dominant landscape forming event being the Alleghanian orogeny. This orogeny was caused by to collision of Africa and North America in the formation of Pangaea. Erosion then followed leading to the gentler topography present today (Bartholomew et al., 2010).

More specifically, the Pittsburgh coal seam is part of the Monongahela group composed of sandstone, limestone, dolomite and coal. The Pittsburgh coal seam was deposited in the late Carboniferous era and the Monongahela group specifically was deposited on an aggrading and prograding coastal plain within the foreland basin adjacent to the Alleghanian fold and thrust belt. Channel sands are present and are thought to have been controlled by deep, early Cambrian basement faults reactivated during the Alleghanian orogeny (Tewalt et al., 2001).

The Herrin coal seam is in the Illinois basin coal field. Krause et al. (1979) wrote a paper specifically focusing on the geology and stability of the roof strata above this seam. They found there were two distinct types of roof rock above the Herrin seam: shaley limestone with faulting and clay dikes or gray shale with roof hazards such as coal or carbonaceous partings. Significant jointing was found in the dark grey shale unit, while clay dike faults were very prevalent in the shale limestone. Krause et al. (1979)
believe most of the structural features formed during the initial stages of sediment
diagenesis and compaction (Krause et al., 1979). Commonly the geologic patterns in
the roof are complex and locally variable (Krause et al., 1979). Many of the geological
features can be explained by differential compaction and it is thought the structural
features formed before the sediments were fully compacted and lithified. The Illinois
basin itself is a broad, gentle structural depression which has seen multiple subsidence
events and marginal uplifts. The Pre-Cambrian basement is overlain by sedimentary
rocks and the New Madrid rift complex (in the Cambrian period) led to initial rapid
deposition of sediment. Once the rifting stopped, the region slowly began to sink and a
shallow sea formed (Kolata and Nelson 1990). Tectonic forces gently folded the
bedrock leading to the formation of the asymmetric Illinois Basin, and the basin is
surrounded by several arches and domes which were active at different times
throughout the history of the basin. There are several basin-wide faults present in the
area (Macke, 2006).

Another coal seam sampled was the Lower Elkhorn bed in the Eastern Kentucky
coal field in the Appalachian basin. The Eastern Kentucky coal field occurs in the
Central Appalachian basin, while the Western Kentucky coal field occurs in the Illinois
basin (Greb et al., 2002). The Lower Kittanning Coal Seam and the Upper Kittanning
seam are also sampled multiple times in the database and are both located in the
Appalachian Basin.

Of all the exposures at Eastern coal seams sampled in the CMRR reference
database, the majority of them are in the Appalachian Basin or the Illinois Basin. From
the above information, it would seem that the geological conditions at the exposures in
the Appalachian Basin are similar to those at Mine A. The Appalachian Basin and the basin containing Mine A are foreland basins with significant uplift, folding and faulting. Both basins have seen considerable erosion too, leading to a relatively gentle, rolling surface topography. It therefore seems that the geological conditions at Mine A closely match those of the mines in the CMRR reference database; this is likely a significant factor in why the CMRR was found to be more effective at Mine A compared to Mine B. The Illinois basin seems to have a similar geological situation to Mine B with a gentle fold across the entire basin. However, the Illinois basin coal seam sampled in the database has notable faulting, and geologic patterns in the roof which are complex and locally variable (Krause et al., 1979). It therefore seems that the geological conditions and the sharp relief of the surface topography at Mine B are not fully captured in the CMRR reference database. This could be because the mines sampled in the CMRR reference database do not fully represent the wide range of possible geologic conditions in the United States. If this is the case, the CMRR would not be expected to be relevant at Mine B as is shown in this work, or indeed for any mine which is in a different geological setting to the mines in the reference database. It is also possible that the geological, and more specifically, the topographical conditions at Mine A are anomalous and are present at very few other coal mines across the US. This idea is discussed in more detail in the following chapter.

6.8 Mine B as a Topographic Anomaly

The possibility of Mine B being overlain by anomalous topography also cannot be discounted. Woolsey et al. (1917) noted the topography of the region to be “peculiar”
based on the differential erosion in the alternating beds of resistant sandstone and clay shale. Aside from a description of the topography and how it stands out from the surrounding region as flat plateaus and sharp river valleys, no other mention of this “peculiarity” is made in the literature. The geological history and structure of the area surrounding Mine B does not seem that extraordinary, rather it is simply more straightforward than that at Mine A. The more unusual topography, and the dramatic changes in slope in the ‘sharp river valleys’ could however make the effects of shear stress more pronounced at Mine B, compared to Mine A. It may also be unusual topographically compared to many other mines. However, since the specific mines used in the CMRR reference database are unknown, the topography above them cannot be evaluated, meaning a more detailed comparison cannot be made here.

The analyses run evaluating the effect of location beneath the base of a valley returned relatively low \( p \) values (0.21 and 0.33 for mines A and B respectively). As only a small number of data location sites were located at the base of a valley, the statistical significance of this correlation cannot be discounted, but it also cannot be concluded here. The evaluation of topographic curvature with roof stability scale returned relatively high \( p \) values (0.34 from the ANOVA analysis and 0.55 from the Spearman test), but the results of the logistic regression analysis show topographic curvature to have a significant impact on the probability of roof stability category at Mine B. This would imply topographic curvature has a significant effect on roof stability at Mine B, however this alone does not conclude whether Mine B is overlain by anomalous topography.
CHAPTER 7
SUMMARY, CONCLUSIONS AND RECOMMENDATIONS
FOR ADDITIONAL RESEARCH

7.1 Research Summary

An analysis was performed to evaluate which non-CMRR parameters are related to roof instability at Mines A and B. An evaluation was also carried out to determine whether the CMRR is applicable to these mines, and if not, potential reasons why were considered. The goals of this research were to use the two case study mines as a starting point for evaluating the effectiveness of the CMRR in predicting roof instability in underground coal mines, in particular in the Western US.

The data set collected at both mines included the component parameters of the CMRR, a record of roof stability and a series of non-CMRR parameters also thought to be indicative of roof stability at these mines. These data were then analyzed for correlation between the CMRR and roof stability, correlation between roof stability and the non-CMRR parameters collected and correlation between the CMRR and the non-CMRR parameters collected. The geological histories of Mines A and B were analyzed to evaluate why these two case study mines present different results. The stratigraphy and geologic history of the mines used in the development of the CMRR system were also examined to answer the question of whether the case study mines used in this research are different from those used in the development of the CMRR.
7.2 Research Conclusions

- The CMRR was found to have a statistically significant relationship with roof stability at Mine A, but not at Mine B.

- The CMRR was found to have greater predictive capability at Mine A compared to Mine B; even at Mine A, however, the appropriate ground stability category (1-4) could not be predicted with greater than 70% certainty using CMRR alone for almost all the locations considered (from the logistic regression analysis).

- The parameters currently considered by the CMRR do not fully explain roof instability at Mine A or Mine B.

- At Mine A, faulting was found to be indicative of roof stability. Depth of cover and surface topography also showed some correlation with roof stability, although the existence of a correlation was not supported by all the statistical tests considered.

- At Mine B, locations at an intersection showed a statistically significant correlation with roof stability. Slope angle, topographic curvature and depth of cover also showed statistical correlation with roof stability to a lesser extent.

- Topographic curvature, slope angle and depth of cover were found to have reasonable predictive capability for assessment of roof stability category at Mine B from the logistic regression analysis.

- Location at an intersection was found to correlate with the roof stability at both case study mines. The CMRR does not currently account for excavation geometry (e.g. span) in any way, so further work could include examining the
effect of the type (3 way/4 way) and span of intersections on roof stability and how this parameter could be combined with CMRR for roof stability prediction.

- At Mine A, the discontinuity cohesion and roughness parameter was the most important component of the CMRR with regard to the prediction of roof stability. This is thought to be due to the high incidence of faulting at Mine A and the statistically significant relationship between faulting and roof stability.

- At Mine B, the discontinuity spacing and persistence rating was found to be the least important component of the CMRR with regard to the prediction of roof stability. This is thought to be partly due to the relative lack of discontinuities at this mine and the frequent presence of massive sandstone in the immediate roof. It could also be a spurious correlation as discussed in Chapter 6.3.2.

- The unusual topographic relief at Mine B, with differential erosion having formed high plateaus and sharp river valleys, is thought to have made the effects of in-situ stress and stress re-distribution on roof stability more pronounced at Mine B, compared to Mine A.

- Combining the potentially increased effects of stress at Mine B with a lack of discontinuities such as slickensides, which are central to the CMRR, means it is not surprising that the CMRR is less applicable at Mine B compared to Mine A.

- The geological location and conditions at Mine A are relatively similar to those in the Appalachian Basin where the majority of the mines in the CMRR database are located – both are foreland basins which have significant faulting and folding present.
Given the lack of information on the topography at the mines in the CMRR reference database and elsewhere in the US, it cannot be said whether Mine B is overlain by anomalous topography or not. Its geological history and setting seems to be similar to the Illinois Basin, however the unusual topography generated by differential erosion at Mine B may be unique among US mines along with the ground control problems associated. There also seems to be a higher incidence of faulting in the Illinois basin compared to the basin containing Mine B.

7.3 Concluding Remarks

The CMRR reference database (Molinda and Mark, 1994) includes generally only one and at most four exposures in each of the coal seams sampled. Further, only previous roof falls or overcasts were used to calculate the CMRR so the locations sampled were predetermined to a certain extent. However, it does seem Molinda and Mark (1994) captured representative samples across the coal fields of the Appalachian and Illinois basin, but not the Western US.

From the discussion in Chapter 6.6, it would therefore seem that the geological conditions at Mine A do match those in the Appalachian Basin relatively well and therefore also match the exposures in the CMRR reference database relatively well. The regional geologic conditions at Mine B seem broadly similar to those of the Illinois Basin, however the local conditions at the coal seam sampled (the Herrin coal seam), are opposite to those observed at Mine B. This suggests Mine B does not match the characteristics of the mines used in the development of the CMRR. This could be due to
several factors. It seems likely that the topography above Mine B has led to a more complex stress field in the roof, although the analyses of the effect of topography on roof stability have not been conclusive in this study. When the magnified effects of stress at Mine B are combined with the lack of discontinuities which are a major focus of the CMRR, it makes sense that the CMRR is less effective there.

This potential variability between mines makes it difficult to have a ‘one size fits all’ approach. However, the CMRR does have a place as a field-based empirical approach, giving an initial indication of roof stability and allowing for comparison between different locations. Making the CMRR system more comprehensive and applicable to Western US coal mines remains a worthwhile goal. It has the potential to be a very useful tool in the prediction of roof fall in underground coal mines and could decrease the number of injuries and lost days even further. A greater understanding of the factors impacting roof stability will certainly benefit the industry and likely lead to more efficient and proactive mining practice as a result.

7.4 Practical Recommendations

- The relevance of the CMRR at a given mine should always be considered before the value is used to inform a roof support plan or make any design calculations.
- The CMRR appears to be more applicable to mines with a high incidence of localized faulting and/or other local discontinuities.
- Where there are few localized discontinuities and/or the topographic relief is steep and varied (e.g. deep, steep valleys and high plateaus), the CMRR may
not be applicable and alternative methods for predicting roof stability may be required.

- One CMRR value is unlikely to be representative of an entire mine.
- Several CMRR values should be calculated at a mine for different geological or geotechnical domains; for each domain considered, a range of CMRR values should be provided in addition to the average value.
- In conjunction with the CMRR, the following parameters should also be considered as potential controls on roof instability.
  - Location at an intersection (excavation span)
  - Topographic Slope (Slope Angle)
  - Topographic Curvature
  - Depth of Cover

7.5 Recommendations for Future Work

A notable limitation of the CMRR is that a single value can very rarely satisfactorily describe the roof stability in a mine as a whole. For example, at Mine A, the values collected ranged from 16 to 47; these two CMRR values would imply significantly different roof stability and thus significantly different support requirements. This means that reporting a single value for the mine is unrepresentative. One possible solution to this would be if the CMRR data could be collected at every panel – this would likely need to be automated but could potentially be done simultaneously with roof bolting as this would allow a large amount of data to be collected with little extra effort from mine personnel. Once a mine had this data, they would have much greater
accuracy in using the CMRR to inform their support plan. For example, if the CMRR was between 25 and 35, they could have a base level of support, if it was from 35 to 45, another category of support could be used, and so on. If this could be done, the CMRR really could become a highly useful tool at each mine, assuming it was an accurate measure of roof stability at the given mine.

Since location at an intersection was found to be strongly correlated with roof stability at both case study mines, future work should evaluate how the type (3 way, 4 way etc.) and span of intersection interacts with the CMRR to influence roof stability. A much larger data set including tens to hundreds of mines would be required to support any such research, however.

At Mine A in particular, a plot of faults and discontinuities on a map along with locations known falls could be made. Contour intervals of known CMRR readings could then be made. This could further examine the idea that the CMRR is more sensitive to geologic discontinuities. Similarly, at Mine B, a map of horizontal stresses could be overlain with CMRR contours and locations of known roof fall. This would allow a visual examination of how roof stability at Mine B is related to the stress conditions.

As discussed in Chapter 6.4, future work could also include the effects of time on roof stability and the calculated CMRR values at a given area. An adjustment factor accounting for time effects could potentially be added to the CMRR. A more comprehensive examination of how the different loading conditions at the data collection sites affect the roof stability and the corresponding CMRR values would also be beneficial. This study also only considers longwall mines; future work including mines
using other underground mining methods particularly in the Western US could allow for a more comprehensive analysis of the CMRR.

If this research is to be continued in the same direction, with a view to making recommendations for changing or updating the existing CMRR system, a much larger data set, incorporating tens to hundreds of mines would be needed to support any such recommendations. A larger sample of Western coal mines would also need to be taken to ascertain if the topography overlying Mine B is anomalous in the US as whole. A larger data set and an accurate method of underground stress measurement would also be required to fully investigate whether the CMRR should be developed to include stress related parameters.

Further research could also consider the relationship between stress-related parameters, CMRR, and roof stability in greater detail. For example, an elastic stress model (considering topographic influences) could be used to estimate horizontal stresses throughout a given mine area. These stress estimates could then be overlaid with CMRR observations and locations of known roof fall.

The Roof Fall Risk Index (RFRI) developed by NIOSH primarily for stone mines could also merit further investigation into its applicability for underground coal mines. It may be that elements of the RFRI could be incorporated into the CMRR should NIOSH decide to further pursue rockmass classification techniques as a method of mine roof stability prediction.
REFERENCES


Freeman J.V., Campbell, M.J., 2007. The analysis of categorical data: Fisher's Exact Test. The University of Sheffield, UK.


The attached file is an electronic copy of the roof support map used at Mine B. To evaluate whether the support used at Mine B was a proxy for roof stability and highlight how much of a confounding influence pre-installed support could be when evaluating roof stability, the non-CMRR parameters collected at Mine B were analyzed with support category. This was only done at Mine B because there were a series of systematic categories of support used and a map showing which support was used where was available. The four ground control categories used to determine support requirements are summarized in Table 4.9 and are presented in greater detail in the attached map which was provided courtesy of Signal Peak Energy.

<table>
<thead>
<tr>
<th>Roof Support Map for Mine B</th>
<th>Single file consisting of a map of Mine B and detailed criteria on the roof support used.</th>
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<tbody>
<tr>
<td>PRIMARY ROOF SUPPORT PLAN MINE B.pdf</td>
<td>PDF file showing detailed descriptions of each of the roof support categories used at Mine B.</td>
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